Electrochemical and Metallurgical Industry

VOL. V. NEW YORK, AUGUST, 1907. No. 8.

Electrochemical and Metallurgical Industry

With which is Incorporated Iron and Steel Magazine

Published on the first of each month by the
ELECTROCHEMICAL PUBLISHING COMPANY,
[*Incorporated.*]
114-118 LIBERTY STREET, NEW YORK.

J. W. RICHARDS, PH. D.....*President*
E. F. ROEBER, PH. D.....*Editor and Secretary*
C. E. WHITTLESEY.....*Treasurer*

Subscription Price: $2 per year, postpaid to any part of the world. Single copies 25 cents.

Copyrighted, 1907, by the Electrochemical Publishing Co.

Entered as Second-Class Matter, June, 1903, at the Post-Office at New York, N. Y., under the Act of Congress, March 3, 1879.

NEW YORK, AUGUST, 1907.

CONTENTS.

EDITORIAL 297
The Metal Market in the Hot Months..... 297
New Zinc Works..... 297
Electric Heat Versus Heat from Fuel..... 298
Symbolism of Chemical Equations..... 298
American Electrochemical Society..... 300
The Iron and Steel Market..... 300
CORRESPONDENCE:
The Townsend Cell for Sodium Chloride Electrolysis. By L. H. Baekeland 301
The Gayley Dry-Blast Process. By Oskar Nagel..... 302
Pyritic Smelting. By Robert Sticht 302
Ontario Meeting of the American Institute of Mining Engineers... 303
Metallurgical Calculations. (Pyritic Smelting). By J. W. Richards. 304
A Burner for Cutting Metals. By M. U. Schoop..... 308
Electrolytic Refining of Tin. By Otto Steiner..... 309
The Iron and Steel Institute Meeting..... 312
Electrolytic Refining of Bismuth. By Arnold Mohn..... 314
NOTES ON ELECTROCHEMISTRY AND METALLURGY IN GREAT BRITAIN 315
SYNOPSIS OF CURRENT PERIODICAL LITERATURE..... 318
ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS..... 321
RECENT METALLURGICAL PATENTS 327
Lead-Lined Hose 331
Cold Galvanizing 331
Hard-Lead Exhaust Fans..... 333
Automatic Still 334
NOTES 334
DIGEST OF UNITED STATES PATENTS. (Calcium Carbide.) 336
NEW BOOKS 337
BOOK REVIEWS:
A. H. Allen's Commercial Organic Analysis..... 337
A. S. McAllister's Alternating-Current Motors..... 337
O. P. Watt's Silicides and Borides..... 338
W. Pick's Ferrosilicium 340

The Metal Markets In the Hot Months.

The metal markets at the opening of the second half of the year do not exhibit the strength and buoyancy this year that they had in 1905 and in 1906. Nor is this so surprising. For the past year we have seen an increase in price as well as in production. These were conjointly caused by an increased consumption. Now it is obvious that consumption cannot long keep increasing at an increasing rate, for this would force prices to so high a level that consumption would be automatically cut off. This is precisely what has happened to some degree in the past six months and accounts for the sagging metal markets. For instance, our net exports of copper are now practically nil. Another allied reason is that there is not enough capital to pay for new construction when labor and supplies are more than a third higher than in 1904. As there is at any one time a fixed amount of capital the increase of cost of supplies by such a practice has diminished the "availability" of capital by just such a fraction. The easing off of the prices of all metals is a sign of a healthy reaction, of a return to a more stable and equitable basis for industry. While 25-cent copper would not make us all pay 10 cents for trolley fare, it would, nevertheless, restrict the number of new electric railroads and retard the electrification of trunk lines of existing steam railroads which have a high density of traffic. There is plenty of money in mining with copper at 15 cents, lead at 5.5 cents, spelter at 6 cents, and pig iron at $15. Higher prices for a long period would cause eventually a restriction of consumption and an expansion of production so great as to cause a bad depression in the metal business when the bubble was pierced.

New Zinc Works.

Conditions in the zinc business point to a large increase in the smelting capacity of the spelter works of the United States. Things usually operate in cycles, and Newton's law of mutual reactions is as true in industry as in physics. The year 1905 we saw a great excess of smelting capacity over ore production. Consequently zinc ore, especially from the Joplin field, was at a premium. The average price for "Joplin 60 per cent concentrates" showed during that year with the average price of spelter at St. Louis a margin of profit only for those plants operated very efficiently as regards recovery and treatment costs. Some of the zinc companies also made money by carrying large tonnages of metal over to the late fall, when the price for spelter was higher. In 1906, however, the pendulum started to swing back and the margin for the smelting operation increased rapidly from nothing to $8 or $10 per ton at the end of the year. The consumption of the metal increased so greatly and the importation of ore from Mexico was so large that even several abandoned plants using coal were put in operation. New plants using natural gas as fuel were also started.

This movement of building new plants kept on at an accelerating pace until the present day, when there are four large gas plants partially built (with one in operation) and nearly as many new coal plants in the Illinois field either in partial operation or in course of construction. As a result of this the smelting capacity of the zinc plants of the country will be nominally increased by the end of the year about 25 per cent, and probably 35 per cent by next Spring. However, as it takes time to start a new spelter works the movement in ore prices will be gradual. But the inception of the movement is seen to-day. There are always delays in the shipment of materials, due to railway congestion, and many other causes retard the erection of plants. Furthermore, the supply of workmen trained to labor about a zinc furnace is limited. But making all allowances for these several factors, it cannot be doubted that there will be a large excess of smelting capacity over ore production in the near future. Consequently, the price of Joplin ore is likely to rise to those limits made by metallurgical and commercial conditions of little or no profit to the average zinc plant. While the production of Joplin, Wisconsin, and Rocky Mountains increases, and importations from Australia, Algeria and Mexico will further augment the ore supply, these factors are not great enough to fill the increased demand.

* * *

As it takes time to develop mines and erect concentrating plants, it will be not before the close of 1908 that the smelters have easier conditions, to say nothing of a harvest similar to that of 1906. In the meantime, it would look as if there would be greater profit in the treatment of low-grade ores, sulphides analyzing 35 per cent zinc and carbonates analyzing 30 per cent zinc, for only a few of the plants care at present to treat ores of so low grade of ore. With a high degree of metallurgical efficiency and with the lessened demand for poorer ore, a works would have a nice margin on the so-called "Western" ores, when Joplin prices were prohibitive. In addition, these ores are so cheap that large amounts of them can be stocked without tying up much capital. The market for the metal continues strong, even with increased production. The fact that 60 per cent of the consumption goes into the steel trade, which is exceptionally healthy in tone, gives basic elements of strength to the market. Indeed, we should be surprised if spelter sold for the remainder of the year at less than 6 cents a pound on the average. The increase in the interest in zinc mines and the number of zinc plants now being built show that this fascinating, volatile and flirtatious metal will occupy a more important place on the metallurgical stage in the future than in the past.

Electric Heat Versus Heat From Fuel.

In the Synopsis of Current Literature in our present issue we notice the last reports of our San Francisco contemporary, *The Mining and Scientific Press*, on the progress of the Noble Electric Steel Co.'s work at Héroult-on-the-Pitt, in Shasta County, Cal. As will be remembered from our former notices this undertaking of Mr. H. H. Noble is noteworthy in two respects from an industrial point of view. If successful it will mark the beginning of commercial manufacture of pig iron at the Pacific Coast, and at the same time it is the first commercial plant in the world in which the electric furnace enters into competition with the blast furnace. A concise analysis of the possibilities of electric heat versus heat from fuel seems timely, and it seems proper to first view the problem from the narrow point of view which is most unfavorable to the electric furnace and which simply inquires as to those relative costs of fuel and electrical energy for which the cost of the blast-furnace and of the electric-furnace treatments becomes the same. The immense importance of the element carbon in metallurgy is due to its double application as a strong reducing agent and as a fuel to produce, by its oxidation, the heat needed for the chemical reaction. In the case of reduction of iron from the ore, the blast furnace and the electric furnace require the same quantity of carbon to act as reducing agent and to combine with the oxygen in the iron oxide. But the blast furnace also requires carbon for producing the necessary high temperature, and it is only this last portion of carbon, the calorific value of which is replaced by electrical energy in the electric furnace. We will, therefore, first compare the cost of fuel and electrical energy to produce the same number of heat units.

* * *

One kilowatt-hour is equivalent to 860 kilogram-calories, which is approximately the full heat obtained from 100 grams of good coal when completely burnt to carbon dioxide. If the cost of a ton of coal of 2,000 pounds (or 907 kilograms) is a dollars, and if the efficiency of heat production by burning coal is x per cent, then the coal required for producing 860 kilogram-calories costs $a \div 90.7\ x$ dollars. On the other hand, if the cost of 1 kilowatt-hour is b cents and if the efficiency of producing heat from electrical energy is y per cent, then the kilowatt-hours, required for producing 860 kilogram-calories, cost $b \div y$ dollars. Hence electrical heat will be cheaper than heat produced by combustion of fuel if $a\ y$ is greater than $90.7\ b\ x$. To go further, we need the efficiency figures x and y. By assuming definite figures we introduce, of course, an uncertainty into our comparison, but it will probably be considered fair if for a first approximation we assume a 25 per cent efficiency for fuel heating and a 75 per cent efficiency for the electric furnace. Then we conclude that electrical heat will be cheaper than heat produced from fuel if a is greater than $30.2\ b$, or in words, if the cost of a ton of coal in dollars is more than 30 times the cost of a kilowatt-hour in cents. For instance, to compete with coal at $6.00 per ton, the electric kilowatt-hour would have to cost less than 0.2 cent. This is clear evidence that if the electric furnace did not have other important features it could not compete with fuel heat under ordinary conditions.

* * *

We have purposely made this comparison on the basis of the cost of the ton of coal and of the kilowatt-hour, although it has become customary to use the electric horsepower-year as the unit of electrical energy in such estimates. Then the above condition can be stated in this way: electric heat will be cheaper than fuel heat if the cost of 1 ton of coal is more than one-half the cost of the electric horsepower-year. Stated in this form, the comparison looks more favorable for the electric furnace than stated, as in the preceding paragraph. But in

reality it is not proper to make the comparison on the basis of the horsepower-year, since we thereby implicitly assume that the electric furnace is working continuously every hour all year around, which is in general a decidedly improper assumption.

* * *

It is, of course, clear that the above comparison is exceedingly narrow in that it considers only one single side of the problem, namely, the amount of fuel and electrical energy required to produce the same number of calories. But even in this very respect the above comparison falls short of the truth and does not do justice to the electric furnace. The chief reason is that electrical heating is essentially internal heating, permitting a very high concentration of energy at any point wanted and thus enabling one to produce high temperatures. Fuel heating, on the other hand, is always more or less transmission of heat from the fuel to the charge, and the rate of heat transmission depends on the difference of the two temperatures; this rate decreases rapidly the higher we go up in the temperature. Naturally, with fuel heating we can never obtain any higher temperature than that of the burning fuel itself. There is no corresponding limitation in electric heating. This is the fundamental reason why for all very high temperature processes the electric furnace reigns supreme. This is an established fact, but since electric reduction of iron ore does not fall into this class of processes, we may not further discuss them here, except by pointing out that the possibility of obtaining higher temperatures might be found useful for special cases of iron ores; for instance, with titaniferous slags. It is of greater importance perhaps to refer to the fact established at the Sault Ste Marie experiments that iron ores containing a high sulphur contact can be successfully smelted in the electric furnace so as to get a pig iron containing only a few thousandths of 1 per cent of sulphur. Through the electric furnace certain iron ores will therefore become available for the production of pig iron which heretofore could not be treated economically. This will probably become more important in future years than it seems to be at present.

* * *

But there is another point. With respect to the reducing agent it was stated above that the same amount of carbon as reducing agent is required in the electric furnace as in the blast furnace to rob a certain amount of iron oxide of its oxygen. This is perfectly true, but it would be absolutely wrong to conclude that the electric furnace has therefore no advantage in this respect over the blast furnace. The point is that the blast furnace requires coke, while charcoal or peat coke will do very well in the electric furnace. This point is at the bottom of the whole situation in California. On account of coke not being available, California has not had its own pig iron industry. At the new plant on the Pitt River there is unlimited timber for charcoal making. Viewed from every point the undertaking of Mr. Noble looks promising, especially if we consider that pig iron sells in San Francisco at about $10.00 more than in Pittsburg. Let us hope that the work which was officially inaugurated on our national holiday, the Fourth of July, will turn out commercially successful. It would be a matter of great industrial importance for California and would stimulate similar progressive work in other districts where the conditions appear sufficiently favorable for electric smelting and sufficiently different from the conditions in our present blast furnace centers. In the latter centers electric smelting for reduction of iron from ore is, of course, out of question.

Symbolism of Chemical Equations.

For the average engineer who has not made a specialty of chemistry, the significance of a chemical equation is restricted; it simply indicates to him the relative weights of the different substances involved in the reaction. This is something, but it is by no means everything, since every chemical equation has a much wider significance. In a journal devoted to engineering it seems not improper to say a few words concerning the different ways in which a chemical equation may be interpreted. Different interpretations mean different points of view. The equation is mathematically always the same, but according to the key or the dictionary which we apply, the equation reveals a different view of the reaction for which it stands. The dictionary which we apply to find the relative weights of the different substances is the table of atomic weights. But if any of the substances taking place in the reaction is in the gaseous state and we know the pressure and temperature, we can also find directly the volume from the equation; for this we need as key the constant of Avogadro's rule, 22.42, *i. e.*, the number of cubic meters filled by 1 kilogram-molecule of gas at atmospheric pressure and at 0° C. The same key opens the field of solutions, at least of those solutions for which van't Hoff's theory of the parallelism between gaseous pressure and osmotic pressure in solutions holds good.

* * *

Further, to enter an entirely different field, if the equation involves oxidation and reduction it can represent an electrolytic process. If it does, we find directly from the equation the relation between the quantity of electricity (coulombs or ampere-hours) and the weights of the substances transformed; we need as key the constant of Faraday's law 96,540, *i. e.*, the charge in coulombs of a monovalent gramion. But the symbols in a chemical equation may also be considered as masses endowed with energy; then the ordinary equation does not balance; but with the aid of thermochemical tables as dictionary, we find the heat-balance sheet or energy-balance sheet of the reaction. If in case of an electrolytic process we want to apply Thomson's rule for a first approximation, we get from the energy-balance sheet directly the voltage of the reaction; the key is here 0.0000434, that is the figure with which we have to multiply the gram-calories, involved in a gramatomic monovalent change, to get the volts. Finally, if we want to get the capacity of the reaction for performing work we find it from the equation by means of a free-energy table as dictionary, and from this we can find the exact e. m. f. with the same key which is used in Thomson's rule. The importance of tables of physical and chemical constants is thus manifest; they are just as necessary for the engineer as the dictionary is for making a translation into another language. The universal value of the work of men like Berthelot, who has spent almost a lifetime for the determination of thermochemical data, cannot be overestimated. Nevertheless, much still remains to be done even in this field. Far more remains to be done in the determination of the "free energies" of reactions.

American Electrochemical Society.

At a meeting of the Board of Directors, held in Philadelphia on July 31, the following gentlemen were elected members of the Society: Mr. Franz Roessler, manufacturing chemist (Roessler & Hasslacher Chemical Co.), Perth Amboy, N. J.; Dr. Eugene Haanel, Director of Mines, Department of Mines, Ottawa, Can.; Mr. L. D. Vorce, superintendent Pennsylvania Salt Manufacturing Co., Wyandotte, Mich.; Prof. Herschel C. Parker, Columbia University, New York City; Dr. John W. Brown, Director Research and Battery Laboratory, National Carbon Co., Cleveland, Ohio, and Mr. J. G. Kremers, general superintendent Wisconsin Sugar Co., Milwaukee, Wis..

At the preceding meeting of the Board of Directors several vacancies on the board had been filled by electing Prof. S. A. Tucker, of Columbia University; Mr. F. A. J. FitzGerald, of Niagara Falls, and Mr. F. J. Tone, of Niagara Falls, managers of the Society.

The next meeting of the Society will be held this Autumn in New York City, probably early in October. The New York Section of the Society will be in charge of the arrangements, but the president of the Society, Prof. C. F. Burgess, of the University of Wisconsin, who is at present in the East, also spends very considerable time and efforts to make the coming meeting very attractive. The dates of the meeting will probably be either Oct. 3 to 5 or 10 to 12.

The Iron and Steel Market.

July has been an extremely dull month in all branches of the iron and steel trade. There has been scarcely any new business, and that chiefly for early delivery. A dull midsummer is the rule rather than the exception in the trade, but the last two years furnished exceptions, so that the contrast is rather striking.

In a period devoid of very interesting developments the most important has been the relative firmness which has been maintained despite the dullness. In finished steel products there have been no recessions in price; in pig iron the chief change has been the sudden and complete disappearance of premiums for early delivery, while prices for late deliveries have sagged very slightly. Specifications on old contracts have continued to come in fairly well, and postponement has been asked on the shipment of comparatively little material.

The future of the market is necessarily in doubt. Periods of dullness like this usually hatch out changes, if any are at all likely to come at any time. The mills and furnaces entered the dullness with order books very well filled, and have readily been able to maintain shipments up to their capacity, which is somewhat decreased in the hot weather. They can, as a rule, run for two or three months yet without any decided buying movement. Production of pig iron during the first half of the year was about 13,500,000 gross tons, against 13,700,000 tons in the second half of last year and 13,600,000 tons in the first half of last year. With new furnaces lately completed and others to be completed shortly, production in the second half of this year could reach about 14,500,000 tons, or at an annual rate 2,000,000 tons in excess of that in the first half. Imports of ordinary grades of pig iron have been discontinued; nevertheless, to maintain the pace, consumptive demand in the second half would have to exceed that of the first half by more than a million tons a year. All prospects are that it will be decreased rather than increased. Pig iron is in a sensitive position, prices having advanced to a level quite out of line with finished products. It is hard to see, therefore, how pig iron prices can fail to undergo a material recession. Theoretically they could drop several dollars a ton without endangering the price structure on finished material, because, as just noted, they have been out of line with finished material, but time and again the iron trade has shown that this theory does not work out in practice. With prices of finished material stationary, an advance and a subsequent decline in pig iron always dislodges the price structure of finished material ultimately.

July 15 a strike was inaugurated of dock laborers at the ore docks at Duluth and Two Harbors. The following week the men at the Superior docks came out, thus completely tying up all the docks at the head of the lakes, which are the outlet for all the Mesaba and Vermillion range ores except the relatively small tonnage which goes out on all-rail routes. At the same time many of the miners on the Mesaba range, suddenly organized by the Western Federation of Miners, went out. As all these men had individual contracts the employers refused to make concessions. The Menominee, Gogebic and Marquette rangers, with the docks serving them, are not affected. Some efforts have been made to create the impression that this strike has a greater influence upon the iron trade than is really the case. At this writing the matter is not settled, but it can be said that all computations of loss of ore tonnage—and whatever is lost in dock movement is practically lost to the season, as the mines can hardly accumulate much ore—are based upon the assumption that it was necessary to the iron trade to bring down all the ore which physical conditions in the chain of mining and transportation would permit. A decision to that effect may have been reached before the season opened, but conditions have since changed, and it is now far from certain that any very large gain over last season's movement is necessary. The movement to July 15 indicated that a gain of 6,000,000 tons or more could be made over last season's total of 38,500,000 tons, and this strike could be prolonged for a month or more and still permit a moderate gain over last season, which would probably be sufficient.

Pig Iron.

Premiums for prompt pig iron have totally disappeared. In general deliveries are well taken, but in occasional instances consumers have asked for some delay in deliveries, and in still rarer instances there is complaint that furnaces are trying to ship more than the contract tonnage. This has resulted in an ample supply of prompt iron. There has been a slight decline in quotations for delivery in the fourth quarter, and all deliveries are on a parity, about as follows: f. o. b. Mahoning or Shenango Valley furnace (90 cents higher delivered Pittsburg); Bessemer, $22.50; basic, $22.00; No. 2 foundry, $22.50; forge, $21.75 to $22.00.

Steel.

The United States Steel Corporation continues short of steel, the Carnegie Steel Company still being much behind in deliveries on its outside billet obligations. The steel corporation bought over 50,000 tons of open-hearth and Bessemer billets during the second quarter, and purchases have continued through July. The steel has been applied largely on outside billet contracts, but doubtless the purchases would not have been made with such avidity had there not been danger that the open market would decline to a point below the settlement price on sliding scale contracts which are controlled by the average price of pig iron. Some outside interests can readily sell crude steel; one such interest in July sold 500 tons of Bessemer billets and completed shipment three days after the placing of the order. We quote Bessemer billets, delivered Pittsburg, at $30.00 and open-hearth at $31.50 to $32.00. Sheet bars remain at $31.00, f. o. b. Pittsburg.

Rails.

Placing of important rail contracts is still delayed by the discussion over specifications. The most interesting development is the disclosure that the improvements quietly undertaken at the Edgar Thomson rail mill of the Carnegie Steel Co. involve the casting of ingots of double the cross-section employed for years, making 4-ton instead of 2-ton ingots, and the erection of a heavy stand of rolls to precede the present blooming mill to take care of the larger section. At the same

time a change in methods will be made whereby the ingots will not be allowed to cool so suddenly between the converting department and the soaking pit. It is confidently expected that these changes will make sounder steel and reduce the length of the pipe. Demand for light rails continues good.

FINISHED MATERIAL.

Altogether, contracts for steel for ten lake boats, to be built next season, have been taken by the Carnegie Steel Co., and several others are under negotiation. Many semi-annual contracts with jobbers for plates, shapes and bars have been renewed by the mills. Outside of this business new buying has been very light, but as stated, the mills have ample tonnage on books to carry them for two or three months yet.

Prices of finished steel products are unchanged, and are as follows, f. o. b. Pittsburg:

Structural shapes, $1.70 per hundred pounds for beams and channels, 15 inches and under.

Plates, $1.70 for tank quality.

Merchant steel bars, $1.60, base.

Common iron bars, $1.70, delivered Pittsburg, and $1.60, f. o. b. Pittsburg, for Western delivery.

Sheets, 28 gauge, $2.60 for black and $3.75 for galvanized.

Tin plates, $3.90 for 100-pound cokes.

CORRESPONDENCE

The Townsend Cell for Sodium Chloride Electrolysis.

To the Editor of Electrochemical and Metallurgical Industry:

SIR:—In your last issue you published a letter by Mr. J. R. Crocker, which expresses astonishment and doubt as to the extraordinarily high efficiencies of the Townsend cell. Your correspondent cannot be more astonished than I myself and others were when for the first time we witnessed the unusual performance of this remarkable cell.

Just because results were so unexpectedly high special pains were taken to insure great correctness in the methods employed for determining efficiencies. Observations were continued for months in succession day and night; very perfect ammeters and voltmeters were selected, and the instruments were frequently verified; furthermore, analytical methods were checked by independent observers. For calculation of energy efficiencies we, of course, used the very method described by Mr. Crocker, taking the factor 2.29—or in round numbers 2.3—as figured by the Thomson or thermochemical rule, this being the hypothetical lowest voltage for the decomposition of sodium chloride solutions.

I would respectfully suggest to Mr. Crocker to carefully reread the article which astonishes him so much and he will find that nowhere is there any mention of energy efficiencies as high as 68 per cent, so this requires no further answer.

But energy efficiencies of 55 per cent to 60 per cent are quite high enough; yet such efficiencies are not unusual with the Townsend cell, although they quite naturally excite wonder and surprise to all whose experience has been with other forms of electrolytic cells.

The Townsend cell can be run with quite a strong load on voltages as low as 3.3 to 3.4 volt or even lower, but it has been found more expedient and economical to sacrifice energy efficiency to *capacity* and to run at much higher loads, which, of course, will increase the voltage. But even running at 1 amp. per square inch the voltage remains as low as 4 to 5 volts. This is due to the fact that the distance between anode and cathode scarcely exceeds three-eighths of an inch.

There seems to be somewhere some confusion in the statements of Mr. Crocker where he speaks of the voltage of the curve in Fig. 7, which is the record of a defective cell, and substitutes this voltage instead of the much lower voltage of a normal cell, as described in Fig. 6. I take this opportunity for correcting a serious omission of a footnote under Fig. 6, which has occurred in the printing of the report of my paper: It will be noticed that at some points of the curve the indicated ampere-hour efficiency goes above 100 per cent; this seeming impossibility is easily explained by the fact that in the Townsend cell the liberated sodium metal accumulates upon the surface of the cathode plate, forming a sort of alloy with iron; once in a while this accumulated metal reacts upon the liquid, forming sodium hydroxide. Whenever it occurs the stored up alkali metal increases suddenly the output of sodium hydroxide, and at that moment the efficiency curve may be too high. During the period immediately following, the efficiency curve will be correspondingly too low, but averages will come quite close to 100 per cent. A very convincing confirmation of these high efficiencies was made by the repeated analysis of the chlorine gas, which always showed nearly 100 per cent pure, whenever the ampere-hour efficiencies approached the latter number. Furthermore, the amount of HClO and $HClO_3$ in the anode liquor averaged as low as 100 milligrams per liter.

These records relate to commercial cells of full size, but run separately with good care. Under such conditions, whenever the ampere-hour efficiency sank below 97 per cent, we knew that something was wrong, and after correcting the conditions we were always able to increase the efficiency to about 100 per cent.

In practice, however, where cells are run in sets of about sixty and where conditions are not so favorable and things

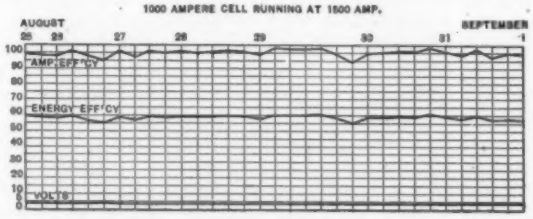


FIG. 6.—EFFICIENCY CURVES.
(Reproduced from page 210 of June issue.)

have to be run rather roughly, a few defective cells may show a depressing influence on the whole system and lower somewhat efficiencies; but even then average ampere-hour efficiencies of 90 per cent are considered very poor work although the current densities were about 1 amp. per square inch of active anode surface. Under good practical conditions, using no special care, 95 per cent to 97 per cent ampere-hour efficiencies are considered normal averages. I know that this statement will astonish many persons who heretofore have considered 90 per cent ampere-hour efficiency very good for other types of cells, although the latter are run at considerably smaller loads.

I ought to mention that lately some disturbance has been caused at the Niagara Falls plant owing to some defective rubber shield insulators, and this has considerably upset the regular operation of the cells; yet under these abnormally defective conditions the average ampere-hour efficiencies have seldom dropped below 90 per cent to 93 per cent. This little trouble is being done away with and will be entirely prevented in the future by a slight modification in this detail of the construction of the cells; the latest reports are 97 per cent average ampere-hour efficiencies for 62 cells.

In answer to the last remark concerning the salt which is carried by the cathode liquor, I desire to state that the amount of salt can be regulated within satisfactory limits by modifying the hydrostatic pressure. Furthermore, the elimination of the salt is an easy operation; as soon as the solution is submitted to evaporation the salt is precipitated, not only by evaporation but also by the fact that sodium chloride becomes unsoluble in concentrated solutions of sodium hydroxide.

I believe Mr. Crocker is mistaken if he thinks that the main corrosive action on steel tubes in vacuum pans is due to the presence of salt. Were it so it would not be practical to manufacture table salt in vacuum pans, yet this method is extensively employed. The corrosive action of cathode liquor occurs mainly when strong caustic lyes are evaporated, and for that reason it has been the practice here and abroad to finish evaporation of strong caustic liquors in cast iron finishing kettles.

L. H. Baekeland.

Yonkers-on-Hudson, N. Y.

The Gayley Dry-Blast Process.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—While traveling in Continental Europe a few months ago, I had several interviews with some prominent iron masters, and I naturally was interested to learn what they think about the Gayley process.

The general impression about this system on the other side is that it does not and will not effect a reasonable saving, *i. e.*, a saving that will pay more than the ordinary interest on the investment. Otherwise, the European iron works would be the first to adopt this system, since the economical conditions over there necessitate the greatest possible economy, especially with respect to raw material, which is high-priced in Europe as compared to America.

If the Gayley process would effect a saving it would have a much greater value (comparatively) in Europe than in America, the cost of the raw materials being a much greater item over there than here. In this country the cost of *labor* is of greatest importance, in Europe the cost of the *raw materials*. Hence, a material saving process which is successful in America will be still more successful in Europe, while a labor-saving process which is successful in Europe will be still more successful in America.

I was informed that in certain iron works the most careful observations and researches have been made, for several years, relative to the influence of the moisture of the air upon the iron output. In some of the iron works the difference of temperature in the summer and winter amounts to 40° C. But even with this great difference practically no increase of the iron output in winter as compared to the summer output was observed. The argument is that since a decrease of 40° C. in temperature means a corresponding decrease of the air moisture, an increase of the yield of the furnace should have been observed, if the Gayley process is to be a success.

Reliable detailed data on the saving effected by the Gailey process would certainly be highly appreciated by iron and steel men.

Oskar Nagel.

New York City.

[From the manner in which the Gayley process has been received in Europe, and especially in Germany, in the technical press as well as in meetings of engineering societies, it would seem that our European friends approach the problem by way of speculating a priori as to what results they may expect from dry blast. In spite of its ingenuity this speculative method which has been so characteristic of certain renowned systems of metaphysics, due to German philosophers, involves great dangers when applied to new engineering problems. In speculating a priori, that is from the cause to the effect, one is liable to overlook that one single cause, the effect of which is to be determined, is often intimately connected with other causes which are also effective. The only proper way to discuss the Gayley process is to take the exact data of its operation in plants where it is used on an industrial scale, and to analyze these figures so as to deduce the causes from the results. This method, which avoids all dangers of speculation and simply analyzes the facts, has been applied to a set of figures of Mr. Gayley by Dr. J. W. Richards, our Vol. III., p. 240. But, of course, such an analysis requires very full experimental data, and we join our correspondent in the hope that full reliable data on the working of the Gayley process—in addition to those already given by Mr. Gayley and Mr. Meissner—will soon be available. We await with special interest the results of the Pottstown plant, to which we referred on page 290 of our last issue. It would also be very interesting if those European iron masters who have observed the influence of moisture of air upon the iron output during a series of years would publish their figures.—Editor.]

Pyritic Smelting.

To the Editor of Electrochemical and Metallurgical Industry:

Sir:—In the "Synopsis of Periodical Literature" of the Electrochemical and Metallurgical Industry of February last, there is a reference, on page 59, to our smelting process here, in which data derived from an article of mine in Borchers' "Metallurgie" of 1906 are compounded with others taken from a paper read by Mr. R. Nicholls before the Chemical, Metallurgical and Mining Society of South Africa. I am not acquainted with the latter paper, but the information given by Mr. Nicholls is evidently greatly obsolete, in some respects, and positively erroneous in others. There are grave contradictions between Mr. Nicholls and myself. Now, these disparities are not of professional interest to me personally, but they may mystify some of your American readers who may take a deeper interest in the subject matter, because, unfortunately, your reviewer erroneously alludes to Mr. Nicholls as a "late assistant manager of the works," thus tacitly ascribing to Mr. Nicholls the possession of a full acquaintance with the subject of his paper in its latest phases.

In view of the share which my own writings have had in the compilation of Prof. Peters' "Principles of Copper Smelting," the unfortunate intrusion of Mr. Nicholls in the guise of a presumptive sharer of my experiences, but with a discrepant presentation of same, is likely to give rise to a puzzled condition of mind as to who is right and who is wrong. For the sake of such readers I beg to state that Mr. Nicholls left the Mount Lyell Co.'s employ for South Africa some four or five years ago, after having filled the position of junior assistant in the chemical laboratory. Prior to his departure his experience in connection with our smelting method was practically nil, or, to be fairer to him, his acquaintance with it was restricted to such facts, of a general nature, as a young man in his position had the opportunity of gathering from observation. It is evidently these crumbs of information that he has written up in South Africa, in itself a sufficiently laudable action, but hampered, in this instance, by inaccuracies on the one hand and on the other by a serious neglect of the circumstance that things did not, since his departure, remain standing at the point where he thought he saw them while here.

I might also, perhaps, take this opportunity of referring to your reviewer's reference (page 31 of your January issue) to a supposed omission on my part, in Borchers' "Metallurgie" of Nov. 8, of the heat of slag formation during the union of ferrous oxide and silica, as if this elementary fact were unknown to me. I had, however, no reason whatever for mentioning it. Your reviewer has evidently not grasped the purport of my simply casual reference to the circumstance that the amount of iron present in the particular sulphide which does the work in the pyrite furnace evolves a greater amount of heat than does the sulphur accompanying it. It cannot be said that whatever additional heat the subsequent union of ferrous oxide and silica may contribute, such increment is due to the burning of the iron. The fact is that the heat of slag formation may with equal justice be ascribed to the silicon, which is no part of the compound originally acting as the fuel, as to the ferrous oxide.

Robert Sticht.

General Manager, Mt. Lyell Mining & Railway Co., Ltd.

Queenstown, Tasmania.

Ontario Meeting of the American Institute of Mining Engineers.

The Ontario meeting of the American Institute of Mining Engineers was held July 23 to 30, opening on the 23d in Toronto. The headquarters were in the palatial King Edward Hotel, the opening session being on the afternoon of Tuesday, the 23d, in the banquet hall of the hotel. The address of Dr. Douglas, on "Some Reflections on Secrecy in the Arts," was the feature of this session, and it is needless to advise those who know Dr. Douglas how he handled this subject and cauterized the ultra-secretive and foolishly shortsighted policy of some individuals and corporations.

The reception in the evening at the Parliament Buildings was a revelation to most of the visitors. The rich splendor of the edifice, combined with the formal and dignified tone given to the occasion, gave the impression of being 3,000 miles away from the United States instead of nearly within sight of our shores. Some of the engineers are said to have whispered to their intimates that the occasion would have been more enjoyable if it had been not quite so stiff. However, the warmth of Canadian hospitality and good friendship beat beneath the courtly bearing and medal-covered breasts of the Institute's hosts, and the oftner the scene is recollected the pleasanter becomes its memory.

Wednesday, the 24th, morning and afternoon, were given over to reading and discussion of papers. The indispensable secretary of the Institute (everybody knows his name) was indefatigable in stirring up discussion and giving point and piquancy to the proceedings. A splendid oil painting of Dr. Raymond, by the way, stood in one corner against the wall, and smiled benignantly on the assemblage, so that even when the Doctor was out of the room his spirit seemed to contemplate and participate in the proceedings. The painting was presented by some one to somebody else, but everybody seemed to know its resting place and its object; it will ornament the Institute's quarters in the Engineering Building, and will there continue for many centuries, let us hope, to dispense the benediction of the Doctor's genial smile and to prolong his memory and his influence in the Institute.

By common consent the paper of the day was Mr. Saunders' account of the electric air rock drill. To be a mining engineer you have to drill and be drilled, and therefore the subject interested everybody. But the conclusion, announced so positively by Mr. Saunders, that compared with the ordinary air drill the power consumption was *halved*, took the meeting by surprise. How tremendously important, if it is true, was the universal sentiment, coupled with the corallary: but it is true, since Mr. Saunders says it.

At 8 P. M. the members of the Institute who could spare the time boarded the special train provided by the liberality of the government, bound for Cobalt, while those who could not go regretfully turned their faces homeward.

* * *

We are afraid some of our Canadian friends are looking for results from this excursion of a kind not to be expected from it. A leading journal informed its readers that Cobalt (also Sudbury, etc.) was going to be *investigated* by the Institute, that on their return they would give their *verdict*, and the exact truth about these regions would be given to the world, thus putting an end to all uncertainty about the value of these deposits, the mines, mining stock, etc., of these regions. All this is purely visionary; no such result can come from the visit; the engineers are on a pleasure trip, they will incidentally give many hints and suggestions we have no doubt, to their Canadian *confreres*, and on their return they will think and act more intelligently than they could have done before in regard to these Canadian enterprises, but they are not going to investigate or settle definitely any such questions as the newspaper misled its readers into believing. The trip will do Canada a healthy lot of good, but there will be something left for many an investigating committee after the engineers are all back home from their trip.

* * *

The program of the trip to the Cobalt district includes visits of Cobalt, New Liskeard, Haileybury, Temagami, with excursions to various mines and trips on the Lake Temiskaming and the Lake Temagami. The veins which have made the area known throughout the mining world were discovered during the building of the Temiskaming & Northern Ontario Railway in the late summer of 1903. The first public announcement of the discovery of the camp was made in November, 1903. In September, 1904, the railway from North Bay had been completed, and the shipments from the camp by the end of the year amounted to 158 tons, valued at $136.217, an average of $862 per ton. The values of the shipments during 1905 and 1906 are shown in the following table:

	1905,		1906,	
	Quantity.	Value.	Quantity.	Value.
Silver, ounces	2,473,452	$1,372,577	5,357,830	$3,543,069
Cobalt, tons	118	100,000	312	30,819
Nickel, tons	75	10,000	156	.....
Arsenic	549	2,693	1,558	.....
Total	.....	$1,485,570	.....	.....

The shipments for 1907 to July 15 amount to about 7,000 tons. The statistics as to value are not complete.

After the visit to the Cobalt district the program provides a trip to Sudbury, the Creighton Mine, the Copper Cliff works of the Canadian Copper Co., the Moose Mountain iron mine and the Vermilion placer deposits.

The Sudbury district is the world's greatest producer of nickel, and the mines and smelting works are equipped with modern machinery.

* * *

The following is the complete list of papers read or taken as read at the meeting:

Some Reflections on Secrecy in the Arts, by James Douglas.

The Wilfley Table, by R. H. Richards.

The Tar Sands of the Athabaska District, by Robert Bell.

The Occurrence of Nickel in Virginia, by Thomas L. Watson.

Condition of the Coal-Briquetting Industry of the United States, by E. W. Parker.

The Temple-Ingersoll Electric Air Rock Drill, by W. L. Saunders.

Notes on (*a*) the Cobalt Mineral Area, by Willet G. Miller; (*b*) the Sudbury Mineral Area, by Alfred E. Barlow (illustrated with lantern views).

Geology of the Virginia Barite Deposits, by Thomas L. Watson.

The Effect of High Litharge in the Crucible Assay for Silver, by Richard W. Lodge.

Physical Factors in the Metallurgical Reduction of Zinc Oxide, by Woolsey McA. Johnson.

Zinc Oxide in Iron Ores and the Effect of Zinc in the Iron Blast Furnace, by John J. Porter.

Chronology of Lead Mining in the United States, by Walter R. Ingalls.

The Promontorio Mine, by Francis C. Lincoln.

Blow-Holes in Steel Ingots, by E. von Maltitz.

The Evergreen Copper Deposit, by Etienne A. Ritter.

Pure Coal as a Basis for the Comparison of Bituminous Coals, by W. F. Wheeler.

Quantitative Field Test for Magnesia Cement Rock and Limestone, by Charles Catlett.

The Production of Converter Matte from Copper Concentrates by Pot Roasting and Smelting, by George A. Packard.

The Panoramic Camera Applied to Photo-Topographic Work, by Chas. W. Wright.

Notes on the Present Source and Uses of Vanadium, by J. Kent Smith.

The Mines of San Juan Depomuceno del Doctor, by T. D. Murphy.

Metallurgical Calculations.

By J. W. Richards, Ph. D.

Professor of Metallurgy in Lehigh University.

Pyritic Smelting.

This method of smelting is sometimes called pyrite smelting, because usually practiced on ores rich in pyrite; it is, however, just as applicable to ores rich in pyrrhotite (magnetic pyrites) or chalcopyrite (copper pyrites), and therefore the more general term *pyritic* smelting is really the more proper and descriptive of the range of the process.

For a full description of pyritic smelting we would refer the reader to the volume "Pyrite Smelting,"[1] a symposium of information contributed by nearly forty metallurgists, and edited by T. A. Rickard or to Sticht's monograph, "Ueber das Wesen des Pyrit Schmelzverfahrens,"[2] or to Dr. Peter's "Principles of Copper Smelting,"[3] which contains a 125-page chapter upon it.

A brief statement of the principles of the process is as follows:. Given an ore containing considerable silica in the free state and a good percentage of pyrite, pyrrhotite or chalcopyrite, it is possible to smelt it down in a shaft furnace to a ferrous silicate slag and a matte by means of cold blast and without using any carbonaceous fuel. Whether fuel is cheap or dear the possibility of dispensing with it when smelting down certain sulphide ores is highly important; yet it is only within a very few years that the principles involved have been well enough understod to make the process a recognized success. As far as *modus operandi* is concerned, the ore is charged into a shaft furnace, water-jacketed at the boshes and smelting zone, with enough flux to furnish 10 to 20 per cent of CaO or similar alkaline earth base to the slag, and a high pressure of blast is employed, such as would correspond to a high rate of driving in ordinary smelting (up to 3½ pounds pressure per square inch is used). The matte and slag either collect in the well of the furnace or run continuously into an external settler, where they separate as in ordinary smelting. The heat necessary to run the furnace is all supplied by the oxidation of sulphur and iron in the furnace, the former escaping as SO^2 in the gases and the latter as silicate of FeO in the slag. The process may be regarded as a very quick partial roasting of the ore, accompanied by simultaneous formation of melted slag and matte, because of the high temperature generated by the oxidation itself.

There is hardly any process in the whole of metallurgy which invites so strongly to quantitative calculations, such as we are endeavoring to encourage by these articles; and there are fewer processes which present such a lack of data upon which to base the calculations. Not only are the physical and chemical (thermochemical) data scarce, but the ordinary industrial data, careful details of the running of the furnaces, weights and compositions of charges and products, analyses and temperature of gases, are very largely lacking—most that we have of value are those furnished as recently as 1906 by Robert Sticht, director of the Mount Lyell furnaces in Tasmania, and these apply only to his particular furnaces and operations.

Fundamental Principles.

Taking the simplest case, and supposing FeS^2 and SiO^2 to be charged into a pyritic smelting furnace, we know without a doubt that the FeS^2 becomes approximately FeS at a red heat, and the sulphur thus evolved is driven off at a part of the furnace where there is no free oxygen, so remains in the furnace gases as sulphur vapor. At a temperature of 1400° Sticht has shown experimentally that the FeS becomes something like Fe^7S^8, and it is known that the sulphur in the matte is often less than can correspond to FeS. But the actual temperature of combustion attained at the moment of oxidation in pyritic smelting is higher even than 1,400°, and it is almost certain that as FeS oxidizes, it is heated so intensely that it really goes through two stages, first decomposing to Fe^3S and then oxidizing to FeO:

$$2FeS = Fe^3S + S$$

$$Fe^3S + SiO^2 + 2O^2 = SO^2 + 2FeO.\ SiO^2$$

or, putting it all together:

$$2FeS + SiO^2 + 2O^2 = S + SO^2 + 2FeO.\ SiO^2$$

If we cast up the thermochemistry of this reaction we find a very considerable evolution of heat, easily comparable with the heat of oxidation of carbon, and accounting for the running of the furnace. The thermochemical analysis of the above reaction is:

	Absorbed.	*Calories.*
Decomposition of 2FeS		48,000
	Evolved.	
Formation of 2FeO	= 131,400	
" " SO^2	= 69,260	
" " 2FeO. SiO^2	= 8,900	
		209,560
Excess of heat evolved =		161,560

The FeS and SiO^2 come into the zone of oxidation already heated to a high temperature by their contact with the rising hot gases. They come to this zone, or focus, heated to at least 1,000°, and perhaps hotter, before they begin themselves to oxidize. The air comes in cool, and we will not credit it with any sensible heat at the moment oxidation begins.

Theoretical Temperature at the Focus.

With these data, and a few reasonable assumptions, we can calculate how hot the focus will become at its hottest point. This calculation is similar to that for the calorific intensity of combustion of a fuel, or the theoretical temperature before the tuyeres of a blast furnace. Making the calculation for the quantities represented by the equation, and assuming

Heat in FeS at 1000° melted	200 Calories.
" " SiO^2 at 1000° solid	260 "
" " Slag ($\frac{5}{3}$ weight of FeO): $0.27\ (t - 1100) + 300$	"
" " Matte ($\frac{1}{3}$ weight of slag): $0.14\ (t - 1000) + 200$	"

We can calculate the theoretical temperature t, to which the slag, matte and gases will be raised by the heat at hand, which latter is the sensible heat in FeS and SiO^2 at, say, 1,000°, plus the heat evolved in the reaction.

	Calories.
Heat in 2FeS at 1000°: 176×200	= 35,200
" " SiO^2 at 1000°: 60×260	= 15,600
" of the reaction	= 161,560
Total heat available =	212,360

Calorific capacity of the products:

SO^2	$= 22.22(0.36t + 0.0003t^2)$
N^2	$= 170(0.303t + 0.000027t^2)$
Slag	$= 240[300 + (t - 1100)0.27]$
Matte	$= 80[200 + (t - 1000)0.14]$
Sum	$= 0.0113t^2 + 135.5t + 5,520 = 212,360$

Whence $t = 1429°$ C.

The result of our calculation is to show that a temperature sufficient to run the furnace is theoretically obtainable if the weight of slag made, carrying a given weight of FeO, is not too great. Supposing the silica and other slag-forming ingredients are so heavy in comparison to the amount of FeO formed by

[1] Engineering and Mining Journal, New York, 1905.
[2] Wilhelm Knapp, Halle, 1906. Reprinted from "Metallurgie."
[3] Hill Publishing Company, New York, 1907.

oxidation that the slag contains only 50 per cent of such FeO, then the weight of slag above would be 288 instead of 240, and the temperature attained only 1,324° instead of 1,429°. This reduction, however, is getting perilously near the temperature necessary to keep the furnace in operation. The formation temperatures of copper blast-furnace slags are 1,100° to 1,200°, and an over-heating of at least 50° is necessary for practical operation of the furnace. Suppose we put 1,200° as the minimum theoretical temperature which will run the furnace, then the maximum weight of slag can be calculated in the above solution for t, and thence the minimum percentage of FeO.

Let X be the maximum weight of slag, for t a minimum of 1,200°. Then we have:

	Calories.
Heat in SO^2 at 1200°	= 19,248
" " N^2 at 1200°	= 68,422
" " Slag at 1200°	= 330X
" " Matte at 1200°	= 18,240
Whence	$105,940 + 330X = 212,360$
and	$X = 323$ kg.

Since this maximum weight of slag must contain the 2FeO = 144 kg. of FeO, the minimum percentage of FeO produced by oxidation and going into slag in pure pyritic smelting, must be

$$144 \div 323 = 0.45 = \mathbf{45} \text{ per cent.}$$

The margin for working pyritic smelting is ordinarily so small that variations in the temperature of the blast and its humidity must exercise a large influence on the running of the furnace. This coincides with experience as far as the heating of the blast is concerned. At La Lustre Smelter, Santa Maria del Oro, Mr. Koch says that "a warm blast of 200° C. is a *sine qua non* with us; it spelled success; cold blast meant failure." This is, however, an extreme position; many other metallurgists working other ores have run entirely with cold blast, but there is no doubt that smelting is easier, faster and more regular when using warm blast. No one has yet tried drying the blast, but there can be no doubt that under some circumstances this would contribute greatly to the regularity of running and would increase the theoretical temperature obtainable at the focus or smelting zone.

Use of Auxiliary Coke.

When the amount of slag-forming materials in the charge is high, pyritic smelting using cold, ordinary blast may become impossible for lack of sufficient calories to melt the slag and matte. In this case, which often occurs, some coke may be used, the combustion of which to CO^2 at the focus increases materially the heat available and the temperature. If the temperature desired is, let us say, 1,400°, carbon at 1,000° can burn to CO^2, yielding CO^2 and N^2 as products, and give a large surplus of heat to the charge at this temperature.

One kilogram of coke, containing 0.9 kg. of carbon, will form 3.3 kg. of CO^2 (1.67 cubic meters) and 6.35 cubic meters of N^2. The heat generated is 7,290, and adding in the heat in hot carbon at 1,000° (342) we have 7,632 Calories generated. But in the products at 1,400° we have:

	Calories.
CO^2 1.67[0.37 + 0.00022(1400)] × 1400 =	1582
N^2 6.35[0.303 + 0.000027(1400)] × 1400 =	3038
	4620

Surplus, to help the fusion

$$7632 - 4620 = 3012 \text{ Calories.}$$

Each kilogram of coke used will therefore melt down, at 1400°,

$$3012 \div 330 = 9 \text{ kg.}$$

of slag, and thus relieves the situation materially. At Mount Lyell, 0.5, 1.0 and 1.5 per cent of coke (reckoned on the charge) is found necessary, according to the nature of the ore worked. From this we have increasing quantities of coke used up to several per cent, according to the exigencies of the case, and we pass by insensible gradations through partial pyritic smelting to ordinary smelting. As the carbon is increased the sulphides get less and less of the oxygen blown in, and therefore the degree of oxidation of sulphides is lower and the concentration poorer. The best concentration, using unroasted sulphide ore, is obtained by pure pyritic smelting, if there are enough sulphides present to generate the requisite heat and temperature.

Rate of Smelting.

In all that precedes it has been assumed that the furnace was driven fast enough. Other things being equal, the harder a furnace is blown the more material can be put through it, and the smaller the heat losses by radiation and conduction when expressed per unit of charge treated or of product obtained. It is, therefore, possible to increase the temperature in the focus simply by harder blowing, and thus to decrease the amount of auxiliary coke needed. A similar effect is produced by heightening the furnace shaft, since this increases the regeneration of heat by the charges, they coming to the hot zone more intensely preheated by the ascending gases. The present tendency in pyritic smelting is undoubtedly to increase the rate of driving, heighten the furnace, and thus decrease the auxiliary coke needed and dispense with hot blast, which is somewhat of a complication.

The possibility of smelting in any manner depends on being able to generate the theoretical temperature necessary for running the furnace, and then keeping the rate of smelting per minute per unit area of the smelting zone as high as practicable. This achieves two practical results, viz.: makes the actual temperature of melted-down slag and matte approximate closer to the theoretical temperature of the focus, and gets the largest tonnage through the furnace. As we gradually increase the smelting rate we increase the temperature of the focus, because of decreased radiation losses; but, on the other hand, we tend to decrease the relative amount of oxidation, because of the decreased time that the charge is subjected to oxidation; there must be a certain rate of driving which will attain the maximum of oxidation, *i. e.*, of concentration, and past which increased tonnage is put through at the cost of decreased concentration.

Problem 107.

W. H. Freeland (*Engineering and Mining Journal*, May 2, 1903), at Isabella, Tenn., smelted Ducktown pyrrhotite ore in a water-jacketed Herreshoff furnace, having a cross-sectional area at the tuyeres of 21.7 square feet. The analyses of the materials used and the products is as follows:

	Charges.			
	Ore.	*Quartz.*	*Slag.*	*Coke.*
Cu	2.744	.....	0.73	.....
Fe	36.519	1.45	39.20	2.30
S	24.848	0.32	1.75	1.58
SiO^2	18.548	96.79	30.90	8.41
CaO	7.294	0.23	8.51	.....
MgO	2.672	.....	2.71	.....
Zn	2.556	.....	2.88	.....
Al^2O^3	0.911	0.32	1.90	3.56
Mn	0.770	.....	0.85	.....
O	.....	0.38	11.37	1.00
C	.....	.....	.....	83.86
CO^2	3.138	.....	.....	.....
Loss	.....	0.39	.....	.....

	Products.		
	Matte.	*Flue Dust.*	*Slag.*
Cu	20.00	2.20	0.37
Fe	47.15	30.80	38.84
S	24.00	16.51	1.74
SiO^2	0.44	23.92	32.60
CaO	0.10	4.45	8.24
MgO	.....	1.38	3.44
Zn	2.05	2.98	1.54

Al^2O^3	0.82	1.94	1.50
Mn	0.53	0.15	0.80
O	4.91	15.26	10.88

The charges and products per 24 hours and per 1,000 of ore used were:

Charges:		
Ore	68.0 tons	1000 lbs.
Quartz	5.4 "	80 "
Slag	9.8 "	145 "
Coke	2.3 "	34 "
Products:		
Matte	8.34 tons	122.65 lbs.
Flue dust	1.75 "	25.71 "
Slag	63.81 "	938.24 "

Blast applied, 4,500 cubic feet displacement per minute, at 17-ounce pressure. Assume temperature of gases 450° C. and that they contain no CO, SO^3 or free O^2 (no analyses are given). Assume matte and slag issuing from the furnace at 1,300° (no temperature given).

Required:

(1) A balance sheet of everything entering and leaving the furnace.

(2) The volume efficiency of the blowing plant.

(3) The heat generated per minute, per square foot of cross-section, in the focus of the furnace.

(4) The theoretical temperature at the focus.

(5) The proportion of the heat generated in the focus by the combustion of carbon and by the oxidation of sulphides.

(6) If hot blast were used what should be its temperature to be able to dispense with the coke charged? Assume that the pressure was increased so as to keep the delivery to the furnace constant per minute.

Per 1000 of Ore Smelted.

Charges.		*Matte.*	*Flue Dust.*	*Slag.*	*Gases.*
Ore (1,000).					
Cu	27.44	24.53	0.57	2.34	.....
Fe	365.19	57.83	7.92	299.44	.....
S	248.48	29.44	4.24	16.33	198.47
SiO^2	185.48	0.54	6.15	178.79	.....
CaO	72.94	0.12	1.14	71.68	.....
MgO	26.72	.....	0.35	26.37	.....
Zn	25.56	2.51	0.77	22.28	.....
Al^2O^3	9.11	1.00	0.50	7.61	.....
Mn	7.70	0.64	0.14	6.92	.....
CO^2	31.38	.....	.....	.....	31.38
Quartz (80).					
Fe	1.16	.....	.....	1.16	.....
S	0.26	.....	.....	.....	0.26
SiO^2	77.43	.....	.....	77.43	.....
CaO	0.18	.....	.....	0.18	.....
Al^2O^3	0.26	.....	.....	0.26	.....
H^2O	0.71	.....	.....	.....	0.71
Slags (145).					
Cu	1.06	.....	.....	1.06	.....
Fe	56.84	.....	.....	56.84	.....
S	2.54	.....	.....	.....	2.54
SiO^2	44.81	.....	.....	44.81	.....
CaO	12.34	.....	.....	12.34	.....
MgO	3.93	.....	.....	3.93	.....
Zn	4.18	.....	.....	4.18	.....
Al^2O^3	2.76	.....	.....	2.76	.....
Mn	1.23	.....	.....	1.23	.....
O	15.31	6.04	3.93	5.34	.....
Coke (34).					
Fe	0.78	.....	.....	0.78	.....
S	0.54	.....	.....	.....	0.54
SiO^2	2.86	.....	.....	2.86	.....
C	28.51	.....	.....	.....	28.51
Al^2O^3	1.21	.....	.....	1.21	.....
O	0.10	.....	.....	0.10	.....
Blast (1,191).					
O^2	274.91	.....	.....	97.92	176.99
N^2	916.37	.....	.....	.....	916.37
	2450	122.65	25.71	946.16	1355.77

Notes on the Balance Sheet.

The oxygen of the blast goes as SO^2 and CO^2 into the gases and as oxides into the slag. The amount required for the gases is

$$O \text{ in } SO^2 = S \times \frac{32}{32} = 100.96$$

$$O \text{ in } CO^2 = C \times \frac{32}{12} = 76.03$$

$$176.99$$

$$O \text{ for Fe} = 330.68 \times \frac{16}{56} = 94.48$$

$$O \text{ for Zn} = 26.46 \times \frac{16}{65} = 6.51$$

$$O \text{ for Mn} = 8.15 \times \frac{16}{55} = 2.37$$

O in slag = 103.36

Contributed by charges = 5.44

Contributed by blast = 97.92

To burn sulphides and carbon at the Focus = 176.99

Total from blast = 274.91

(2) The furnace receives 1,191 pounds of blast per 1,000 of ore smelted. This represents at 0° C.:

$$\frac{1191 \times 16}{1.293} = 14{,}738 \text{ cubic feet.}$$

Per 2000 lbs. ore	= 29,475	"	"		
Per 68 tons ore	= 2,004,300	"	"	per	day
	= 83,510	"	"	"	hour.
	= 1,400	"	"	"	minute.
At 50° C.	= 1,477	"	"	"	"
Blower displacement	= 4,500	"	"	"	"

$$\text{Efficiency of blower} = \frac{1{,}477}{4{,}500} = 0.328 = \mathbf{32.8} \text{ per cent.} \quad (2)$$

[It is high time that practical furnace men should stop giving the mechanical displacement of their blower as the amount of air received by their furnace. They still keep doing that, although the furnace receives only from 85 per cent down to 25 per cent of the given volume. What per cent of the piston displacement the furnace is actually receiving is a highly important datum, but is more often unknown than known to the practical men. It can usually only be found by calculation, as is illustrated in the preceding case. We urge upon practical furnace managers, for their own information and use, to cease being satisfied with *piston displacement*, and to get deeper into the real inwardness of their furnace processes by calculating the *air actually received* by their furnaces. The two things are enough different to make it always "worth while."]

(3) To calculate the heat generated at the focus we will assume that all the fixed carbon of the coke is there burned to CO^2, and that the rest of the oxygen blown in produces the reaction characteristic of pure pyritic smelting. We have treated per minute

$$\frac{68 \times 2000}{1440} = 94.4 \text{ lbs. of ore.}$$

and the carbon burned per 1000 of ore is 28.51, generating

$$28.51 \times 8100 = 230{,}930 \text{ lb. Calories.}$$

this absorbs

$$28.51 \times 8/3 = 76.03 \text{ lbs. of oxygen}$$

leaving $274.91 - 76.03 = 198.88$ lbs. of oxygen to oxidize sulphides.

Since, now, $2O^2$ generates by the pyritic smelting reaction 161,560 calories, we have generated per pound of oxygen thus used:

$$161{,}560 \div 64 = 2{,}524 \text{ lb. Calories.}$$

and per 1000 of ore smelted we will have

$$2{,}524 \times 198.88 = 502{,}000 \text{ lb. Calories.}$$

total heat generated

$$502{,}000 + 230{,}930 = 732{,}930 \text{ lb. Calories.}$$

Since this is per 1000 of ore smelted, per minute we have

$$732{,}930 \div 1000 \times 94.4 = 77{,}640 \text{ lb. Calories.}$$

and per square foot of smelting zone area

$$77{,}640 \div 21.7 = \mathbf{3{,}575} \text{ lb. Calories.} \qquad (3)$$

This last figure is extremely useful in determining the relative activity of different furnaces, and in most cases will be a reliable index of the rate at which the furnace is capable of being driven.

(4) Taking as the basis of calculation 1,000 pounds of ore smelted, there is generated at the focus 732,930 pound Calories, there is used 1,191 pounds of blast, and there arrives at the focus all of the charges except the flue dust, CO^2 of ore, H^2O of quartz, and approximately one-quarter of the sulphur. We, therefore, have arriving at the focus about

28.51 lbs of fixed carbon.
526.50 lbs. of sulphides.
621.20 lbs. of inert slag-forming material.

These, assuming them to reach the focus at 1,000°, would bring back into it the following amounts of heat:

Carbon	$28.51 \times 380 =$	10,834 lb. Calories.
Sulphides, melted	$526.50 \times 200 =$	105,300 " "
Slag-forming material	$621.20 \times 174 =$	108,089 " "
		224,223 " "
Heat generated at the Focus	$=$	732,930 " "
Total heat available at the Focus	$=$	957,153 " "

Letting t be the theoretical temperature at the focus, then we have

Heat in Slag	$946[300 + (t - 1100)0.27]$
" " Matte	$123[200 + (t - 1000)0.14]$
" " S vapor	$76[179 + (t - 445)0.11]$
" " SO^2	$35[0.36t + 0.0003t^2]$
" " CO^2	$53[0.37t + 0.00022t^2]$
" " N^2	$727[0.303t + 0.000027t^2]$.

[The 76 pounds of sulphur vaporized at the focus is the total sulphur charged, less the 25 pounds assumed as vaporized above the focus, and less the 100.96 pounds oxidized at the focus; the expression is the total heat in 1 pound of sulphur vapor at t in pound-Calories. The 70 of SO^2 is the weight of sulphur dioxide formed, in pounds, divided by 2.88; the 53 of CO^2 is the weight of CO^2 formed, in pounds, divided by 1.98; the 727 of N^2 is the weight of N^2 divided by 1.26. To be strictly logical, the weights of SO^2, CO^2 and N^2 should be first multiplied by 16 to get ounces, and the resulting expression in ounce-calories divided by 16 to get pound-Calories. We have simply dispensed with these two numerical operations.]

The sum of the calorific capacity of the products at t° is:

$$20{,}098 + 530.5t + 0.0425t^2 = 957{,}153$$

whence $t = \mathbf{1569}^0$ (4)

It will be noted that the temperature is much higher than could have been attained by attempting to smelt this mixture without coke. If the coke were omitted from the charge we would be without the 28.51 pounds of fixed carbon and some 55 pounds of slag-forming material; the products would be without the CO^2 and the N^2 corresponding to it, and the heat available would be decreased 230,930 pound Calories from the absence of carbon. If these corrections are made, and it is still assumed that the materials come down into the smelting zone at 1,000°, the theoretically calculated temperature is

$$t = 1447^0$$

While this temperature is theoretically sufficient, yet variations in ore quality and temperature and humidity of the air would make running under these conditions much more precarious than with the coke.

(5) This has already been calculated. We found 230,930 pound-Calories to be due to the formation of CO^2 and 502,000 to the oxidation of sulphides, making of the total

31 per cent. from oxidation of carbon
69 per cent. from oxidation of sulphides. (5)

(6) If coke were dispensed with, the theoretical temperature previously attained, 1,569°, might be reached if the air used were heated. With cold blast we have under (4) calculated a theoretical temperature of 1,447°. Under these conditions the heat available was altogether 725,266 pound Calories. But to heat the products, whose heat capacity was

$$20{,}083 + 451.7t + 0.0247t^2$$

to $t = 1{,}569°$ would require (evaluating) 789,600 pound Calories, which is 64,334 pound Calories more than are available. To supply this difference by heating the blast we reckon first the amount of blast per 1.000 of ore smelted, which will be 862 pounds and its heat capacity:

$$\frac{862}{1.293}(0.303t + 0.000027t^2)$$

making this equal to the heat to be supplied, 64,334 lb. Cal. we have

$$202t + 0.01805t^2 = 64{,}334$$

whence $t = \mathbf{313}^0$. (6)

Problem 108.

At Mount Lyell, Tasmania, R. Sticht analyzed the gases at different depths of the shaft in a pyritic smelting furnace, and found them of nearly uniform composition for 6 feet down, leaving out the sulphur vapor, which condensed in taking the samples. At 6 to 7 feet below the top, close to the focus of the furnace, the mean of 5 analysis gave by volume

H^2	0.00
SO^2	0.00
SO^3	7.90
CO^2	3.56
CO	0.00
O^2	0.88
N^2 (difference)	87.66

The CO^2 comes from coke, which it was stated was used to the extent of 1.5 per cent of the charge. The slag contained 53 per cent FeO, 30 per cent SiO^2.

Required:

(1) The percentage of the heat generated in the furnace coming from the oxidation of sulphides and the combustion of carbon.

(2) The heat developed in the furnace per unit of slag formed.

(3) The volume of blast per 1,000 kg. of slag formed.

Solution:

(1) We will assume the reaction of oxidation of sulphides to be

$$FeS + 2O^2 = SO^2 + 2FeO$$

and, in fact, the above analyses are our chief justification for so doing. The air blown in is approximately 20.8 per cent of oxygen by volume, and according to the above equation the oxygen going to form FeO is equal to that going to form SO^2. We have then for the oxygen used

O^2 to form 7.90 SO^2.....	7.90
O^2 to form —— FeO.....	7.90
O^2 to form 3.56 CO^2.....	3.56
O^2 to form 0.88 O^2.....	0.88
Sum	20.24

$$O^2 \text{ corresponding to } N^2\ 87.66 \times \frac{20.8}{79.2} = 23.00$$

There is thus a small deficit of oxygen, but if the sulphide oxidizing were assumed to be FeS the reaction would be

$$2FeS + 3O^2 = 2SO^2 + 2FeO$$

And the oxygen accounted for from the gases would be only

O^2 to form 7.90 SO^2.....	7.90
O^2 to form —— FeO.....	3.95
O^2 to form 3.56 CO^2.....	3.56
O^2 to form 0.88 O^2.....	0.88
	16.29

Which barely accounts for two-thirds of the oxygen which must have accompanied the nitrogen.

The conclusion must therefore be that the sulphide which is being oxidized is Fe^2S, and not FeS, for otherwise the gas analyses would show impossible conditions.

The oxidation of the sulphides gives for each O^2 thus used 161.560 ÷ 2 = 80,780 Calories, and the oxidation of C to CO^2 97.200. From the above analyses we infer that for 3.56 volumes of oxygen used for CO^2 15.80 volumes were used in oxidizing sulphides. The relative amounts of heat thus generated are therefore:

By carbon	97,200 × 3.56 =	346,000 =	**21.3**	per cent.
By sulphides	80,780 × 15.80 =	1,276,300 =	**78.7**	" " (1)

(2) The production by oxidation of 2FeO (144 kilograms) is accompanied by the production of SO^2 (64 kilograms) and the evolution of 161,560 Calories. The slag being 53 per cent FeO, the slag formed by the reaction is 144 ÷ 0.53 = 272 kg. The 64 kg. of SO^2 would be, in volume, 22.22 cubic meters, and would be accompanied in the gases by

$$22.22 \times \frac{3.56}{7.90} = 10 \text{ m}^3 \text{ of } CO^2.$$

which contains 10 × 0.54 = 5.4 kg. of C.
whose heat of oxidation to CO^2 is

5.4 × 8100 =	43,740	Cal.
but, heat from sulphides =	161,560	"
total heat available	205,300	"

per kg. of slag 205,300 ÷ 272 = **755** Cal. (2)

Per unit weight of charge this heat would be slightly lower.

(3) The oxygen used per 272 kg. of slag is

for SO^2	22.22	m^3
" FeO	22.22	"
" CO^2	10.00	"
Total	54.44	"
per kg. of slag	0.20	"
" 1000 kg. of slag	200.	"

Volume of air per 1000 kg. of slag made

200 ÷ 0.208 = **960** cubic meters (3)

The Ohio Copper Co., which was incorporated in Maine a few days ago, and which F. Augustus Heinze of the United Copper Company is to head, was formed to take over the Ohio group of mines in Bingham, Utah, which for some time have been under the control and management of Mr. Heinze, and in which large ore bodies already have been developed. A 4,000-ton mill is being built to concentrate these ores and the first section will be completed in November. It will treat 2,000 tons a day. The second section should be ready by next May. In connection with Mr. Heinze's Utah mining operations it is stated that his Salt Lake representatives already have secured title to a site for his proposed 3,000-ton smelting plant which is to be built by the Miners' Smelting Co. The new smelter will be located about 2 miles west of the old Garfield bathing resort at the south end of the Great Salt Lake.

A Burner for Cutting Metals.

By M. U. Schoop.

A well-known physical experiment is the burning of a white-hot iron wire in pure oxygen, yielding a pyrotechnical effect due to finely-divided iron oxide being thrown off in all directions. An application of this phenomenon—for cutting metals—has, however, been made only during the past few years after success had been obtained in welding iron, copper and other metals autogenously with the oxygen flame, so that iron may be joined directly with iron, copper with copper, etc., without using any foreign metal.

An analogous case is the electrolysis of water which has been known as a laboratory experiment for the past hundred years, but has become of industrial importance only during the last ten years—at least in Continental Europe, where there are now a number of systems which are in use on an industrial scale.

In principle, a burner for cutting metal consists of an ordinary oxygen-acetylene or oxygen-illuminating gas or oxygen-hydrogen burner, such as used for autogenous welding of metals provided with an additional small tube for *separately* supplying oxygen gas for cutting. If, for instance, an iron or steel plate, 1 inch thick, is to be cut by means of such a burner, the plate is preheated with the normal welding flame up to white heat along the line where it is to be cut, and then a strong oxygen gas current is directed along this line. Under the action of the oxygen the autogenous burning of the iron starts immediately.

Since the oxygen supply may easily be directed on any distinct point, as is evident from the sketch, Fig. 1, the burning of the metal is restricted to those points where the oxygen comes in contact with the plate. The "melting line," or cutting line, has a diameter of only a few millimeters with sharp edges.

It makes no difference whether thin or thick iron plates (up to 20 mm.) are to be cut. Of course, with thicker material a correspondingly greater burner is to be used. It also makes no difference whether the hydrogen-oxygen flame is used, or the illuminating gas-oxygen flame, or the acetylene-oxygen flame, or the benzine-oxygen flame.

The least expensive is the acetylene-oxygen flame, especially when thick plates are to be cut, since this gas mixture evolves the greatest number of calories per unit of gas volume. In some cases in practice, however, illuminating gas or benzine may be used instead of gas or acetylene for the sake of convenience.

With respect to the effect the hydrogen-oxygen, illuminating gas-oxygen, benzine-oxygen flames are about equivalent, although, of course, different types of burners must be used for the different gas mixtures[1]. Especially if illuminating gas or

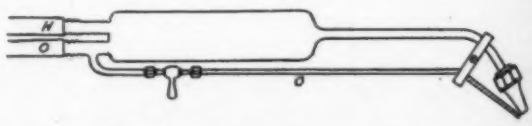


FIG. 1.—BURNER FOR CUTTING METAL.

benzine is used the burner is to be designed according to the suction principle, so that the oxygen gas passing out quickly through the hole sucks the other gas with it. These "injector burners" are advantageously used in all those cases in which two gases are employed of which one is under high pressure (oxygen) and the other under low pressure (benzine or illuminating gas).

To return now to the method of cutting metals by means of such a burner I will summarize the precautions which must be taken.

[1] See also the book of the author Die Industrielle Electrolyse des Wassers (Enke, Stuttgart) and La soudure autogène des Métaux (P. Dusmenil, Paris).

First. The piece of metal to be cut must be placed on two suitable supports in such a way that the melting metal is not prevented from dropping down.

Second. The pressure of the supplementary oxygen-gas supply must be higher the thicker the material to be cut.

Third. The metal to be cut is first heated with the ordinary melting flame and must be brought up to white heat before the separate oxygen-gas current is turned on. The burner should be used as quietly and uniformly as possible and the quicker the thinner the metal to be cut.

Fourth. If one wants to get a very nice and straight cutting line it is useful to make use of a ruler.

Fifth. The burner must be held in such a way that the supplementary oxygen-gas tube almost touches the metallic surface; only under these conditions a pretty sharp cut can be obtained.

As Fig. 1 shows it is not necessary to employ a separate oxygen bomb for the supplementary oxygen-gas tube.

There is no question that such cutting burners may be used to great advantage in many cases in metal industries. On the other hand I must mention one great disadvantage of this apparatus, namely, the possibility of opening by its means within a few minutes the largest "fireproof" safe. This fact should be taken into consideration by manufacturers of safes.

PARIS, FRANCE.

Electrolytic Refining of Tin.

BY OTTO STEINER, PH. D.

HISTORICAL.

While most metals are now being electrolytically refined tin represents an exception. The reason is that in dressing tin ores they are purified to such a degree that the smelting process yields directly very pure tin which does not need any further refining. The small quantities of impure tin which are placed on the market and which have a greater content of foreign metals contain very small amounts of precious metals (½ to 1 per cent Ag). They are used to good advantage for the preparation of alloys.

The first attempt to refine tin electrolytically on a large scale was made by the firm of A. Strauss & Co., in London, in the year 1905, in their smelting works, the Penpoll Tin Smelting Co., Bootle. The process used was invented by Mr. Claus[1], and was somewhat modified by me. About a hundred tons of 90 per cent raw tin were refined, and the product was tin of 99.9 per cent purity.

According to the method of Claus, tin alloys are electrolyzed in a 10 per cent sodium sulphide solution at a temperature of 90° C., with a current density of ½ amp. per square decimeter electrode surface at a voltage below 0.2 volt. Tin is stated to be deposited on the cathode in a pure condition, while the foreign metals (Pb, Sb. Cu, Fe, Bu, Ag, etc.) are precipitated as sulphides and form the anode slimes mostly attached to the anode but partly accumulating on the bottom of the tank.

THEORETICAL.

If a solution of sulphide of sodium is electrolyzed between two tin electrodes, sulphur is set free at the anode and forms tin sulphide, which dissolves in the electrolyte. The sodium ions are discharged at the cathode, and with a sufficiently high voltage the sodium would react with the surrounding water, forming caustic soda and hydrogen, according to the equation $Na_2 + 2H_2O = 2NaOH + H_2$. This would involve the evolution of hydrogen gas; but since the voltage is kept below 0.2 volt no hydrogen gas can be set free, as the voltage is less than the "Haftintensität" of the hydrogen ion. Therefore, the sodium cannot react with the water, but reacts with the surrounding sulphide of tin according to the equation $2Na_2 + SnS_2 \doteq 2Na_2S + Sn$, so that tin is deposited on the cathode and sulphide of sodium regenerated.[2]

[1] Engl. patent No. 297, year 1896.

In this way tin is continuously dissolved and Na_2S consumed at the anode and tin is deposited, and Na_2S is continuously regenerated at the cathode. Since the Na_2S formed at the cathode does not at once mix with the electrolyte, but may be seen to flow down the cathode in streams (Schlieren), care must be taken to mix the electrolyte thoroughly. This is best accomplished by heating the bath from below.

The electrolysis takes place without development of gas, if the voltage at the terminals is kept below 0.2 volt. But as soon as this tension is exceeded, even by a small amount, by increasing the current density or by diminishing the concentration of sodium sulphide in the solution, it rises suddenly and automatically up to 0.6 volt, and violent generation of gas with decomposition of the electrolyte takes place. At the same time the cathodic deposit becomes dull, spongy and contains oxides. But it is in practice of great importance to get dense coherent metallic tin on the cathode, since spongy tin cannot be directly melted and cannot be treated without losses in the furnace.

To obtain always compact, metallic tin deposits, I have found that besides the temperature of the electrolyte, its thorough mixing and the maintenance of a low voltage, the following five conditions are of importance:

(1) Only pure tin plates or tin plated plates must be used as cathodes. Tin deposits on other metals like iron, copper, etc., in spongy form with generation of gas and increase of the tension till up to 2 volts. This is due to the chemical reactions of these metals with the electrolyte ($Fe + Na_2S = FeS + 2NaOH + H_2$) and to the higher potential difference between the changed metals and the electrolyte.

(2) No foreign metals like Fe, Cu, Sb, etc., should be pressed in the electrolyte, either suspended or dissolved or in form of a colloid. These metals have in alkaline solution a lower positive potential than tin, and consequently they are deposited before tin on the cathode surface, which is rendered thereby unsuitable to receive a dense tin deposit. Cathodes which have been spoiled in this way must be removed and replaced by new tin cathodes with bright metallic surface, in order to get good tin deposits. On the other hand, the deposition of the foreign metals before the tin has the advantage that the electrolyte is purified. It may thus be kept in good condition for months so that it is only necessary to add from time to time some sulphide of sodium to keep the original concentration of 10 per cent Na_2S constant. In order to avoid completely the suspension of foreign metals in the electrolyte, it is advantageous to carry out the electrolysis in such a way that the anode slime on the bottom of the bath and that attached to the anodes is not disturbed. Any violent movement of the electrolyte must therefore be avoided.

A circulation of the solution, such as used in the electrolytic refining of copper, is impossible for tin refining. Every tank must be operated independently, and is only in electric connection with the other tanks.

If, for example, after completed electrolysis the electrodes are to be taken out, it is first necessary to allow the solution to flow off very slowly through a syphon-like pipe, the end of which is slightly above the bottom of the cell and above the slime layer; after that the electrodes, and finally the anode slime are removed. The new electrodes may then be inserted and the clear and, if necessary, filtered solution is run into the cell.

(3) Before suspending new tin anodes with metallic surfaces in cell, it is necesary to dissolve in the electrolyte about 1 per cent of its weight of sulphur, preferably in form of

[2] The electrolysis can also be explained in another way as follows: The SnS_2 formed at the anode dissolves in the electrolyte, forming sulphostannate, according to the equation (1) $SnS_2 + Na_2S = SnS_3Na_2$, while the sodium ion reacts at the cathode with the surrounding water (2) $Na_2 + 2H_2O = 2NaOH + H_2$. By keeping the electrode potential below the Haftintensität of the hydrogen ion, the hydrogen is prevented from being set free and decomposes the SnS_3Na_2 into H_2S, Na_2S and tin, the latter being deposited on the cathode. The equation is (3) $2H_2 + Na_2SnS_3 = 2H_2S + Na_2S + Sn$. The hydrogen sulphide, formed according to equation (3), and the NaOH, formed according to (2), will react and regenerate Na_2S, the equation being (4) $H_2S + 2NaOH = Na_2S + 2H_2O$.

flowers of sulphur. The sulphur forms sulphide at the anode, and since it acts as a depolarizer it is possible to start the electrolysis with low voltage. If no flowers of sulphur are added before new anodes are introduced the raw tin anode behaves as an insoluble anode, and as soon as the circuit is closed the voltage rises to about 0.6 volt, with violent evolution of gas and decomposition of the electrolyte, but without deposition of tin. The cathodes get dull and are no longer suitable for tin deposition. But it is possible to use the same electrolyte repeatedly for electrolysis, if every time before introducing new anodes flowers of sulphur or sulphur in other forms are added to the bath. The electrolyte may thus be kept in good condition for several months.

(4) Since the electrodes are arranged in this process according to the multiple system, and since the tension (0.1 to 0.18 volts), and, therefore, the resistance are very small, it is difficult to distribute the electric current uniformly over all electrodes of a bath. The slightest irregularity in the resistance of the contacts greatly influences the distribution of the current, and therefore the uniform working of the electrodes. It is necessary to bring all anodes and all cathodes of a bath to the same electrical potential. This can be accomplished in a very simple manner by using sufficiently large cross-sections for the copper bar and for the electrode contacts, so that their electrical resistance is practically negligible. By directly soldering the electrodes to the homogeneously tinned conductor bars, it is possible to obtain perfectly uniform contacts in a simple and rapid manner, so that the uniform working of the electrodes is assured. It is also advantageous to connect all parallel electrodes of a bath together by soldering with tin strips of sufficient thickness.

(5) It is of importance not to interrupt the current during electrolysis, since otherwise the polarization current generates gas at the anodes and oxidizes the cathodes and fouls the electrolyte.

The first experiments on a large scale were made by the inventor, Mr. Claus, for the firm of A. Strauss & Co. in the year 1902, in Birkenhead (England). Four lead-lined wooden tanks were used, which were connected in series. Each tank was 2.4 meters long, 70 cm. wide and 90 cm. deep. In each of these tanks 30 anodes of 93 per cent Peru tin were arranged according to the multiple system, 70 cm. long, 55 cm. broad and 1 cm. thick, and the same number of tinned-iron-sheet cathodes of about the same size was provided. The electrolyte used was a 10 per cent commercial sodium sulphide solution of 1.070 specific gravity.

The dynamo produced a current of 1,000 amps. at 1 volt, so that every tank consumed about 0.25 volt. The temperature of the elcerolyte was kept between 80° and 90° C. The baths were heated indirectly by steam, by means of lead coils placed on the bottom of the tanks.

After a fortnight of continuous electrolysis, the anodes were consumed about one-half, and on the cathodes a very beautiful and compact metallic tin deposit was obtained which was melted and yielded about 1 ton of tin of 99.9 per cent purity.

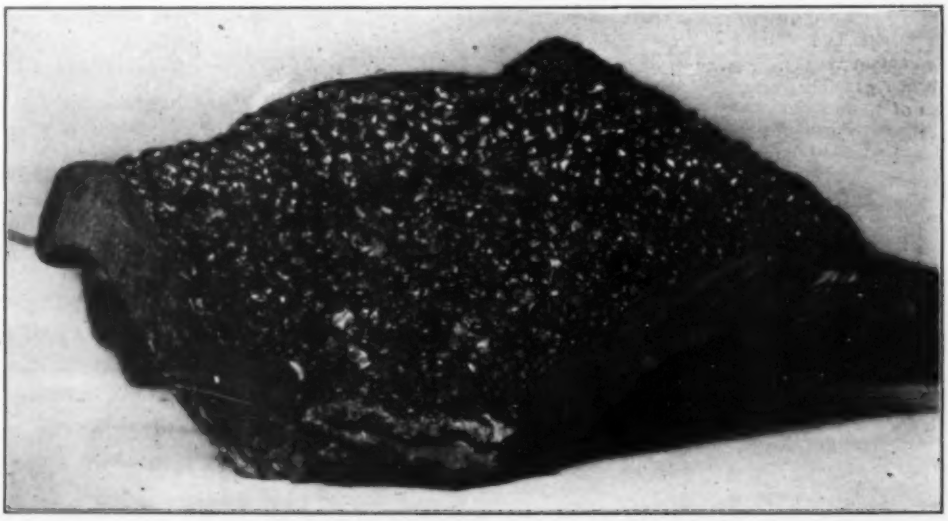
All foreign metals, all silver and all gold and also some tin remained in the anode slime. The experiment was not continued, but the establishment of a large plant for a yearly production of 1,000 tons pure tin was decided upon.

The plant was erected in Bootle, near Liverpool, in 1905. It consisted of 100 tanks, a Dowson gas producer, a gas engine and a dynamo for 1,500 amps. and 25 volts. The tanks were 3 meters long and otherwise arranged in the same way as in the preliminary experiments.

In starting the operation, it was first tried to use simple iron-sheet cathodes. But it was impossible to obtain on such cathodes a coherent metallic tin deposit, since the tension always rose to 2 volts.

Claus, who superintended the work up to this time, died in August, 1905, without having seen the final success of the endeavor of twenty years of his life. In September of the same year I was placed in charge of the work. After replacing the iron cathodes by pure tin cathodes and by following strictly the other conditions of the Birkenhead experiment. I succeeded without difficulties in getting a solid metallic tin deposit.

But when I tried to use the same solution for a second run the voltage increased suddenly up to 0.6 volt and violent generation of gas at the electrodes took place. The reason was that the commercial sodium sulphide contained originally a small amount of poly-sulphide, which acted as depolarizer during the first run and was therefore consumed, so that it was no longer available for the second run.



ELECTROLYTIC TIN.

(Photographic reproduction of a sample piece of tin sent by the author. The piece is bent to show thickness. The cathode plates are, of course, straight and flat.)

By dissolving 1 per cent flowers of sulphur in the electrolyte each time when new anodes were suspended and by strictly following the above conditions (1) to (5), it was possible to use the same solution as often as desired. Twenty-five tanks were then set to work, which produced about 100 tons of pure tin during six months.

Method of Working.

The electrolyte is preferably 10 per cent Na_2S solution, obtained by dissolving the 60 per cent commercial Na_2S in the

necessary quantity of water, settling and, if necessary, filtering. Solutions with higher contents of Na_2S may also be employed. They have the advantage to permit the use of higher current densities and to shorten therefore the duration of the electrolysis.

During electrolysis some of the sodium sulphide is always lost by oxidation of the electrolyte and by formation of sulphostannate. It is therefore necessary to test the solution from time to time and to add some Na_2S. For each 100 kg. of electrolytic tin produced an addition of about 10 to 15 kg. Na_2S (counted as Na_2S) is sufficient. The amount of tin in form of sulphostannate in the solution increases continuously during electrolysis, because more tin dissolves from the anode than is deposited on the cathode. At the cathode there is therefore a very slight generation of hydrogen gas, but this is hardly noticeable. The amount of tin in the electrolyte reaches a maximum of about 2.2 per cent Sn after three months. It does not increase any more after it has reached this amount, but remains constant. If the solution cannot be used any more the tin may be precipitated from it by means of sulphuric acid as tin sulphide, and is then converted into SnO_2 by roasting and finally reduced to metallic tin.

The heating of the electrolyte is best accomplished indirectly by steam. If lead coils are used for heating, the pipes should be of small diameter (exterior diameter not more than 4 cm.), since pipes of larger diameter are easily destroyed by strong rushes of water when the steam is turned on. To save heating expenses, it is advantageous to cover each tank with a wooden cover and to immerse the electrodes below the surface of the electrolyte whereby the evaporation is diminished, and to use the waste steam of the steam engine. The evaporated water is to be replaced by continuous new supply.

The anodes are cast in iron moulds from raw tin containing not less than 90 per cent tin, because the anode slimes remain partly attached to the anodes and increase the voltage too rapidly if the anodes are too impure. This increase of voltage is the quicker the more impure the anodes are, so that tin anodes with about 85 per cent of tin can only be consumed to half the thickness of those of 90 per cent tin. If, therefore, metal with less than 90 per cent tin are to be refined it is better to mix this metal with purer brands in order to bring the mixture up to 90 per cent. Material containing a large amount of iron, say, more than 2 per cent, is not suitable for anodes.

If these anodes are about 75 cm. long, 50 cm. wide and 1 cm. thick, their weight will be about 30 kg., and the workingmen will be able to handle them without difficulty. A greater thickness of the anodes would not be economical, since it would only increase the weight of the residue and the loss of interest on tin. The cathodes are cast as very thin plates from pure tin from the previous operation. The weight of a cathode is about 8 kg.

The ampere-hour efficiency is nearly theoretical, if short circuits between anode and cathodes are avoided. The temperature of the electrolyte should be at least 80° C. The higher the temperature the better the deposit. The cathode deposit is shown in Fig. 1. If I am not mistaken it is the first time that metallic and compact tin deposits of any desired thickness have thus been obtained electrolytically.

The deposit could be still improved by adding a colloid to the electrolyte, for example, gelatine as used in the Betts process for lead refining.

The current density is about ½ amp. per square decimeter anodic or cathodic surface. In order to use a higher current density it is necessary to increase the amount of Na_2S in the electrolyte accordingly as the voltage is increased. The distance between the electrodes is preferably made as small as possible, about 3½ cm.

The tanks are preferably made from iron, since wood gets deformed at a temperature of 90° C. They should not be longer than 2½ meters, since otherwise it becomes difficult to distribute the current uniformly over all electrodes of a bath; the conductor bars would have to be made very thick.

The voltage between the electrodes is about 0.1 to 0.18 volt, and the voltage of the whole bath is about 0.2 to 0.28 volt if a current of about 1,300 amps. is used. As source of power, steam engines are more suitable than gas engines, because steam engines are not liable to cause so many stoppages (see above condition No. 5), and the waste steam may be used for heating the electrolyte.

The management of the process is as follows: Each tank is operated independently of the other baths. First, the anodes and the straightened cathodes are suspended in the bath, the contacts are soldered and the clear and, if necessary, filtered, solution is allowed to flow in. Then the steam is turned on and at the proper temperature the circuit is closed.

At the beginning the electrode voltage is about 0.1 volt. It increases continuously and will be 0.16 volt after a fortnight. The positive and negative leads of the tank are then short circuited, and the electrolyte is allowed to flow out very slowly by means of a syphon, so that the anode slime is not stirred up.

While the tank is slowly being emptied the electrodes are washed at the same time by the vapors rising from the surface of the electrolyte, so that further washing of the electrodes, involving losses of electrolyte, is unnecessary.

When the bath is empty, the electrodes and the anode mud are removed, new electrodes are introduced, the contacts are made, the electrolyte to which the necessary quantity of sodium sulphide and of sulphur has been added is allowed to flow in, the steam is turned on, the short circuit is removed and the electrolysis is started again. The cathodes, which are taken out of the bath may be directly melted, whereby a metal containing 99.9 per cent tin results. The anode slime is removed from the anode residues by scraping with a T-piece and the remaining residues are melted in the furnace.

It is preferable to first roast them in order to remove the sulphur. The anode slime contains in dry state about 12 to 15 per cent Sn, 20 to 30 per cent Sb, 20 to 30 per cent Pb, ½ per cent Bi, ½ to 1 per cent (150 to 200 oz.) Ag, 1 to 2 per cent Fe, 1 per cent Cu, 0.001 per cent (0.25 oz.) Au. The other parts are sodium, sulphur and oxygen. The different metals may be obtained from the slime by the known methods. In washed and dried condition its weight increases about 10 per cent on heating in air; this proves that most of the metals are in metallic condition. The anode slime on the outside contains less tin (12 per cent) than that in the interior of the residue (27 per cent tin).

With a current of 1,200 amps. we obtained from every bath during a fortnight's electrolysis from 500 kg. 93 per cent Peruvian tin about 370 kg. 99.9 per cent pure tin, 100 kg. anode slime, 16 kg. cathode dross and 80 kg. anode dross, while 9 kg. tin remained in the solution as sulphostannate.

It is to be remarked that the anodes are not completely consumed, but that about 50 per cent of the tin remains in the residue. It is, therefore, necessary to use for the above electrolysis about 1,000 kg. 93 per cent anode tin, from which 500 kg. will remain as residues and will be melted after finished electrolysis. There are also necessary about 300 kg. pure tin for the starting cathodes, so that together 1,300 kg. tin must be suspended in the bath to produce 370 kg. of pure tin.

The cost of the plant for a yearly production of thousand tons pure tin will be about £5,000.

The expenses of the treatment for producing 1 ton (= 1,000 kg.) of pure electrolytic tin with such a plant are in shillings:

100 kg. sodium sulphide (counted as 100 per cent Na_2S)	25
25 kg. flowers of sulphur	3
Wages	32
Heating of the electrolyte	10
Cost of electrical energy	14
Loss of interest on tin stored in the process (5 per cent)	30
Repairs	3
Regeneration of tin from the electrolyte	8
Amortization (10 per cent)	10
	135

There are to be added at least 40 shillings for losses of tin in the electrolyte, treatment of the by-products and treatment of the anode slime. so that the expenses per ton electrolytic tin will be about 175 shillings. On the other side there is a gain from winning the silver and other metals contained in 1 ton of raw tin as follows:

	Shillings.
30 to 40 kg. of lead.....	14
30 to 40 kg. antimony.....	70
20 to 30 oz. silver.....	50
¼ oz. gold.....	2
½ to 1 kg. bismuth.....	10
	146

We must keep in mind that the treatment of the anode slime offers great difficulties on account of the great amount of tin it contains, which will involve great losses of metals and make the treatment very expensive, so that the gain will not be 146 but perhaps only 115 shillings.

The electrolytic tin refining process will, therefore, only pay if the raw tin can be obtained at a much lower price than the market price of pure tin. In the year 1905 the price for 1 ton of 90 per cent raw tin was about £112 to £116 on the basis of £130 for pure tin. In 1 ton of 90 per cent raw tin are contained 900 kg. pure tin, £117; the gain from the foreign metals, £5; total, £122; expenses for treatment, £7½; remaining, £114 10 shillings.

It is seen that the profit to be expected is very small. On the other hand, a great risk is run in this process, since the price of tin is very unsteady, and because for a yearly production of 1,000 tons of tin a stock f 150 to 200 tons tin is necessary.

But the greatest, nearly invincible difficulty for this process will be the getting of the raw material, as the whole yearly production of 85 to 93 per cent tin that is put on the market amounts to only about 1,000 tons and smaller plants will work much more uneconomically.

Crefeld, Germany.

The Iron and Steel Institute Meeting.

In continuation of the abstracts, published in our last issue, we herewith give abstracts of those papers which were presented but not read. For an account of the proceedings the reader is referred to the regular monthly letter of our London correspondent.

Carbon-Tungsten Steels.—Mr. Thomas Swindon's paper describes a research having for its object an investigation of the influence of varying percentages of carbon in the presence of a constant percentage (3 per cent) of tungsten carried out in the metallurgical department of Sheffield University. The paper is somewhat lengthy, covering thirty-five pages of printed matter and illustrated by fourteen plates of diagrams and microphotographs. The author's main conclusions are summarized as follows:

1. A definite alloy of iron and tungsten, having the empirical formula Fe_3W is formed on melting the two elements together in sufficient proportion.

2. The 3 per cent of tungsten has decidedly raised the tenacity of the steels without materially lessening the ductility. The elastic ratio is much higher than for carbon steels. A maximum tenacity is seen at 0.9 per cent carbon, but brittle hardness continues beyond this, as shown by alternating stress tests.

3. Below a certain initial temperature all the steels examined show critical points as for carbon steels.

In steels below 0.35 per cent carbon, heating beyond this initial temperature lowers Ar 1 to a definite "low point"; Ar 2 is practically unaffected; Ar 3 is gradually suppressed in its normal position, but appears again below Ar 2, forming the upper maximum of the low point.

With carbon contents 0.35 per cent to 0.9 per cent Ar 1 is first lowered, and then increased heating displaces Ar 3.2 towards Ar 1. Beyond 0.9 per cent carbon the Ar 3 2.1 point is lowered as a whole by heating beyond the lowering temperature, and produces a single low point.

As the carbon increases higher initial temperature is needed for the lowering to be effected.

The "low point" is practically constant at 570° for 3 per cent tungsten steels.

Rate of cooling does not affect the position of the low point after lowering has once taken place.

The range of recalescence is not merely widened, but the critical point is at a distinct lower temperature.

At Ac 1, on heating, a change of similar nature to the carbon change occurs; and at the lowering temperature further change in constitution takes place, the reverse of which is at the low point.

No recalescence has been found at the lowering temperature.

4 Microscopic examination confirms the cooling curves, showing the eutectic composition practically unaltered. The structures resemble carbon steels, but are finer in pattern.

Sentinel Pyrometers and Their Application to the Annealing, Hardening and General Heat Treatment of Tool Steel, by H. Brearly and F. Colin Moorwood.—This paper consists of an essay on pyrometry in general, and the use of sentinel pyrometers in particular, composed of metallic salts with easily observed and reliable melting and solification points, such as sodium chloride (770° to 775° C.), potassium chloride (740° C.), barium chloride (930° C.). These are cast into small cylinders and placed in small porcelain saucers upon the floor of the furnace or oven to be controlled. On the temperature limit being reached the cylinders soften and melt down into a clear liquid in the saucer. Mixtures of two salts can be used.

The Distribution of Sulphur in Metal Ingot Moulds, by J. Henderson.—This note discusses the segregation of sulphur, and shows that the very high sulphur is concentrated in the central of the metal at the top and diminishes very considerably towards the sides. It is, therefore, urged that in sampling such a mould at the top, without taking into consideration the proportion of the top to the rest of the mould, the results obtained, as regards sulphur, must be very unreliable, and consequently of no value. The mould in question measured 7 feet long and weighed 4¾ tons. A section 2 inches deep cut from the top would weigh 270 pounds. The average composition of that 2 inches is estimated from the data available to be practically 0.095 per cent sulphur. Taking the remainder of the mould to be 0.043 per cent, the average for the whole is 0.044 per cent sulphur, a percentage with which there is no reason to find fault, as regards the use of the moulds, when scrapped, for melting purposes in the open-hearth process. The author suggests that the bottom alone should be drilled, and if the sample so obtained be satisfactory the mould should be passed. This is a matter of vital importance to ingot-mould makers, who are often asked by their customers to supply moulds with a maximum of 0.050 per cent sulphur. As the segregation of the sulphur in the top of the mould is scarcely under their control it is certainly not just to them to sample the top unless in due proportion to the rest of the mould.

Abstracts of Reports on Research Work Carried Out by Holders of Carnegie Research Scholarships.

(I.) Copper Steels, by Pierre Breuil.—The results of the tensile tests obtained show that copper increases the tenacity but does not diminish the ductility of steel, although the proportion to which it has this effect varies with the treatment which the metal has undergone. Annealing renders these steels more similar to each other, whereas quenching intensifies their differences. Low coefficients of tensile strength are never encountered with these copper steels, which from this point of view equal in value nickel steels containing corresponding proportions of nickel.

Shock tests on notched bars have shown that, generally speaking, copper steels are somewhat more brittle than steels not containing copper. This does not, however, imply that they cannot be used, as their resilience is also exceedingly high.

Shock tests on plain bars lead practically to the same conclusion as the preceding tests, but are more trustworthy. The steel containing 0.16 per cent of carbon and 4 per cent of copper is remarkable by reason of its resistance to shock. Copper steels are not more brittle than steels containing the same percentage of nickel.

Torsion tests were carried out only on the ingot steel. The results agree closely with those of th etensile tests. The elastic limit, the maximum tensile strength, and the angle of torsion on breaking, are in harmony with the corresponding figures obtained in the tensile tests.

The hardness of these steels agrees fairly closely with their tensile strength. It is usually greater to those of nickel steels containing the same percentage of carbon and the same amount of nickel when subjected to the same thermal treatment.

Corrosion tests were carried out both on the ingot metal and on rolled bars. Only those steels which are employed commercially—that is to say, containing not more than 4 per cent of copper—were subjected to these tests. The corrosive bath consisted of sulphuric acid at 66° Baumé and of water in equal volumes. It was found:

1. That the ingot metal corroded much more rapidly than the rolled metal.

2. That the loss in weight by corrosion diminished in proportion as the percentage of copper rose.

3. That there were certain anomalous results obtained.

4. That it is particularly in the case of low-carbon steels that the difference between steels containing copper and those without makes itself felt.

All the steels up to 4 per cent of copper are capable of being forged hot without cracking.

The conclusions arrived at by the author as to the micrography of copper steels agreed with those of Mr. J. E. Stead. When over 4 per cent of copper is present, reddish nodules containing a high percentage of copper separate in the ingots, this phenomenon being more marked in proportion as the steel contains a higher percentage of carbon. The chief characteristic of the copper steels which are commercially usable (that is to say, not exceeding 4 per cent of copper) is the fineness of their structure; they contain a large proportion of granular pearlite (or sorbite) in proportion as the percentage of copper is higher. This sorbite affords homogeneity, tensile strength and hardness to the metal, and makes it approximate in character to a more highly carburized steel. The copper in dissolving in the ferrite still leaves this constituent considerable malleability, and as a result of this dissolution, the carbon being enabled to form a carbide, finally divided, and in greater abundance than usual, exerts a special hardening influence in consequence. The resulting metal is harder than steel containing a higher percentage of carbon and less copper and at the same time is less brittle.

From the point of view of the nature of the constituents these steels offer no particular difference from ordinary steels; but the form, distribution and quantity of these constituents are such that the special properties indicated above single them out for the attention of the metallurgists in the same way as steels containing nickel or other elements which are usually more expensive than copper.

(II.) Cast Iron as Cast and Heat Treated, by W. H. Hatfield.—The author's conclusions are:

1. That there is undoubtedly occasionally a great variation in the strength of cast irons of the same composition as cast. 2. That this variation does not appear to follow any distinct rule with regard to the temperature of the casting operations. 3. That a difference in mechanical tests is generally accompanied by a difference in the microstructure. 4. That the inequalities of the metal can be rectified by judicious heat treatment, *i. e.*, the irregularity need not persist after heat treatment, at any rate, under certain conditions.

Here, it is felt, the subject must be left for future research to explain the present lack of correlation between the results of the published experiments upon the influence of casting temperature upon the strength of cast iron as cast and heat treated.

(III.) The Genesis of the Lapland Iron Ore Deposits, by Otto Stutzer.—This report is purely geological, and its ultimate conclusions are as follows: The phosphoric magnetite deposits of North Sweden are all associated with plutonic rocks of the syenite family; they have been formed in a magmatic manner, and, indeed, either as magmatic separations *in situ*, or as perigrinating magmatic separations (magmatic veins and bedded streams). Pneumatolysis plays no inconsiderable rôle in the formation of these ores.

(IV.) The Non-Metallic Impurities of Steel, by E. F. Law.—The impurities have been considered as consisting of five in number, namely, iron sulphide, manganese sulphide, iron silicate, manganese silicate and iron oxide (or manganese oxide).

Iron sulphide very rarely occurs in commercial steels, and is therefore not considered at length.

Manganese sulphide is always present in steel, and is usually harmless. The only instance in which it has been found to exert injurious influence on the quality of the steel is when it segregates with phosphide of iron in the form of "ghosts."

Silicates of manganese and iron are frequently found in steel, and are highly injurious to the quality of the metal. In large forgings they sometimes occur in considerable masses, but in rolled steel they are distributed throughout the mass. In either case they are responsible for many failures. No indication of their presence is afforded by ordinary commercial means, and they can only be detected under the microscope.

Oxide of iron frequently occurs in Bessemer steel. It occurs in a finely divided state, and there is evidence that it is soluble in steel. Methods for the determination of oxygen in steel are given, together with the results obtained in practice. As a general rule, steels which on pickling evince a tendency to blistering are high in oxygen. The effect of hydrogen on steel containing oxide is discussed, and experiments are given with a view to determining the temperature at which the oxide is reduced. The results of these experiments tend to show that the oxide is reduced at 100° C. Iron oxide differs from other impurities present, in its electrical behavior, and the influence of this difference on the corrosion of iron and steel is discussed. It has been found that the presence of oxide accelerates corrosion, and corrosion of welded iron affords an illustration of this action. Other instances of the effects of the presence of oxide be found in the pitting of boiler plates and tubes.

The paper ends with suggestions as to the advisability of taking into consideration the influence of impurities, more especially of silicates and oxides, and the effect of the latter on the corrosion of iron and steel.

The Nomenclature of Iron and Steel.—The Council of the Iron and Steel Institute have circulated for consideration the report of the committee of the International Association for Testing Materials. Some forty-four definitions are submitted, and in conclusion the view is expressed that it would be well to decide on a definite carbon content to serve as a boundary line between ingot iron and ingot steel, between puddled iron and puddled steel, and between any other varieties of wrought iron and weld steel. Two plans have been considered. One is to draw this line at 0.32 per cent carbon or its equivalent in other elements, for the reason that this carbon content appears to correspond to the critical point O in the diagrams of Roberts-Austen and Roozeboom. This has the merit of corresponding to a definite physical boundary. The other plan is to draw the boundary at 0.20 per cent of carbon, because this is a convenient place to separate the important classes "soft steel" and "half-

hard steel," so that if this point was adopted "ingot iron" would be synonymous with "soft steel," and "ingot steel" would be equivalent of the two classes "half-hard steel" and "hard steel."

The Congress have requested the committee to continue their labors with a view to securing the foreign equivalents of the terms defined.

Electrolytic Refining of Bismuth.

By Dr. Arnold Mohn.

Very little has been published concerning the electrolytic refining of bismuth containing precious metals. The following quotation is from Prof. W. Borchers' German book "Elektrometallurgie":

"There is no difficulty in parting bismuth containing precius metals in acid aqueous solutions of the more easily soluble salts of bismuth. Although the bismuth is thereby not deposited in a well cohering condition, this is no great disadvantage for the electrolytic refining of bismuth, since the

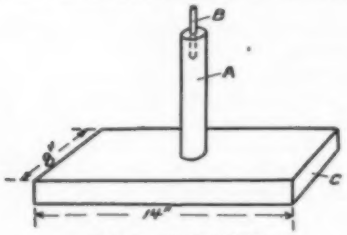


FIG. I.—CAST ANODE.

smelting of metallic bismuth in powdered form offers comparatively little difficulty. The precious metals remain, of course, at the anodes. In view of the relatively small quantities which will be treated by such a process, stoneware vessels are mostly used as electrolytic cells."

A recent patent by Mr. Anson G. Betts relates to the electrolytic refining of silver containing bismuth with methyl sulphuric acid as electrolyte.

I had an opportunity to become practically acquainted with the electrolytic refining of bismuth, the object being the parting of a Mexican lead-bismuth bullion of the following composition:

	Per Cent.
Pb	81.10
Bi	14.50
Sb	1.58
Fe	1.46
Zn	0.43
As	0.29
Ag, Au	0.40
Cu, Co, Ni	traces

It was intended to recover the lead, bismuth, silver and gold. The process used was a combination of electrolytic and metallurgical processes, namely:

First—Electrolytic refining of lead by the Betts process.

Second—Purification of the slimes.

Third—Electrolytic refining of bismuth.

Fourth—Electrolytic refining of silver and gold.

Anodes cast from the bullion are electrolyzed in a solution of 6 per cent $PbSiF_6$, 14 per cent H_2SiF_6 (total acid). The lead deposited on thin cast cathodes is quite pure. It contains only about 0.01 per cent of Bi. In the anodes which retain their original form there remain Bi, Sb, As, Cu, Ag and Au. I will not give here a description of the Betts process, since it has already been described in detail in this journal, but I will say that in the case of which I speak it was very successful.

The cathodes are melted and cast into ingots; if they contain any Bi, Sb and Sn it is removed by poling.

The anode slime is scraped off and partly directly broken off from the thin middle layer of the anode. This work is easy, but on account of the danger from hydrofluoric acid the workingmen must always wear rubber gloves. An analysis of these slimes give

	Per Cent.
Bi	83.92
Sb	6.80
Pb	4.50
Ag	2.48
Cu	1.70
Insoluble SiO_2 and Au	0.50

After having been removed from the anodes the slimes are placed on a movable lead-lined filter box, are washed with hot water, and after having been dried they are mixed with caustic soda and soda ash and then treated in a reverberatory furnace until all As, Pb, Sb are removed. For removing copper an addition of sodium sulphide is used.

Samples are taken, and if a sample gives large leaves and yields a clear solution in concentrated nitric acid, the firing is stopped and the casting of the bismuth anodes begins. These anodes consist of rectangular plates, the thickness being 1¼ inches, the width 8 inches and the length 14 inches, with a round rod or candle in the center for connection to the electric circuit.

These round rods or candles are separately cast. Two halves of a 2½-inch iron tube, held together by means of clamps, are mounted on a wooden block and serve as mould.

On account of the brittleness of the alloy a copper pin is cast into the head of the candle to make the connection with the cable. When the anode plate is cast the candle is simply placed in the center of the mould. Fig. 1 shows a cast anode. A is the candle, B the copper pin and C the anode plate proper.

The electrolyzing apparatus is arranged in the same way as described in Balbach's United States patent 588,524 for parting silver and gold. Twelve stoneware tanks are used as electrolyzing vessels. They are placed lengthwise in two rows and rest on brick.

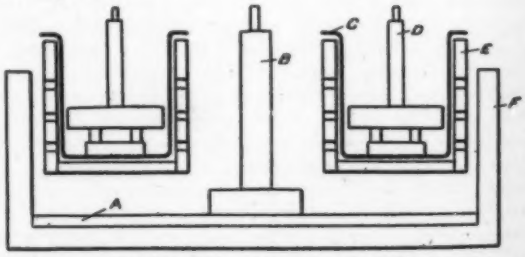


FIG. 2.—VERTICAL CROSS-SECTION OF ELECTROLYTIC CELL.

The anode boxes with lattice bottoms and perforated longitudinal walls are suspended in pairs across any electrolyzing cell. These filtering boxes are lined with a filtering cloth on which a small wooden frame is placed. The anodes rest on this frame. The small wooden frame is necessary so that the electrolyte can circulate directly below the anodes.

Acheson graphite plates are used as cathodes. They are placed directly on the bottom of the cells. It is unnecessary to provide a special joint between the plates, since in time bismuth accumulates firmly in the crevices and thus forms the joints. For the connection to the outside electric circuit a heavy bismuth candle is used which rests on the graphite plates.

Fig. 2 shows a vertical section of the electrolytic cell and Fig .3 a plan. In Fig. 2 A is the graphite cathode, B the bismuth candle resting on the graphite, D the anode, E the filter box and C the filter cloth. F is the electrolytic tank.

In order to prevent consumption of the anode candles their ends are covered with paraffine, so that they may be used

several times, a new anode plate being simply cast onto them.

The analysis of anodes gave the following results:

	Per Cent.
Bi	94.0
Pb	2.20
Ag	3.13
Cu	0.48
Sb	0.11
Au	0.07

The electrolyte used is a solution of bismuth chloride and free hydrochloric acid. It is made by dissolving pure bismuth in nitric acid, precipitating the bismuth as oxy-chloride, washing the precipitate and finally dissolving it in HCl and diluting it to a concentration of 7 per cent Bi and 9 or 10 per cent HCl.

The electrolysis is carried out with a cathodic current density of 20 amps. per square foot, while the current density at the anodes is three times this amount. The voltage at the terminal of the cell is 1.2.

Of course, with this high-current density, all possible impurities pass into solution together with the bismuth. From the electrolytic potential series it is seen that the metals, as far as they do not form insoluble chlorides, pass into solution in the following order: Pb, Bi, Sb, Cu, Ag and Au.

Lead dissolves first. Its concentration in the electrolyte is about 0.1 per cent and crystals of lead chloride are floating on the surface of the bath. Part of the lead goes into the bismuth deposit, so does Sb. The concentration of Cu in the

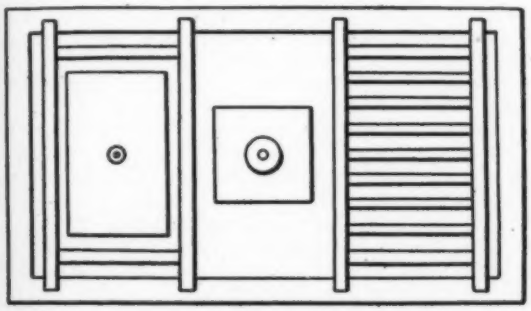


FIG. 3.—PLAN OF ELECTROLYTIC CELL.

electrolyte may be up to 0.2 per cent without depositing this metal to any extent together with bismuth.

The greatest difficulty in the electrolysis comes from the silver. It is intended that it should remain unattacked, or as silver chloride together with the gold in the anode filter boxes. But as a matter of fact after a few weeks silver is found in the bismuth deposit, and the silver content amounts up to 0.3 per cent.

By using good filters, two linen cloths with an intermediate layer of filter paper, it was shown that this high content of silver in the bismuth cannot be solely due to mechanical transfer of the silver to the cathode.

The real reason is the solubility of AgCl in $BiCl_3$. When the silver is in solution it is deposited before the bismuth, as is clear from he potential series.

To keep the silver away from bismuth, there is only one possibility, namely, the diminution of the anodic current density.

The bismuth is deposited on the graphite cathodes, and especially below the anode filter boxes in crystalline form, in form of trees or in beautiful mushroom form, and after the filter box has been taken out the bismuth deposit is removed by means of a wooden ladle. There is often formed on the cathodes a firm metallic deposit which is broken off by means of a chisel.

For a given current density it is important to maintain the concentration of the electrolyte constant, according to the results of the analysis. This is accomplished by adding either $BiCl_3$ or HCl.

If the concentration in bismuth decreases too much the bismuth is no longer deposited in gray crystalline form, but as a black voluminous mass. It then fills the intermediate space between the anode filter boxes and the graphite cathodes, and thus causes trouble.

If there is not sufficient free acid present, bismuth oxy-chloride is formed at the anodes and the voltage rises rapidly.

The bismuth obtained is washed in filter boxes with hot water and then melted at a moderate temperature in plumbago crucibles and the slag is skimmed off.

If the metal has not the proper appearance and the right color it is melted with nitrate and caustic soda until all traces of Pb, As and Sb are removed. A metal of about 99.8 per cent purity is thus obtained. The chief impurity is silver.

The slime which remains in the anode filter boxes and which consists of Ag and Au, besides Sb and Bi, is boiled with dilute HCl until all bismuth oxy-chloride is in solution; it is then filtered, washed and dried, and is then melted with sodium carbonate and cast into anodes for the separation of silver and gold.

For the silver cells the same arrangement of Balbach with anode filter boxes and graphite cathodes is used and the parting of the metals takes place, as is generally known, in $AgNO_3$ and HNO_3 as electrolyte.

From the wash-water, due to the washing of bismuth and of the bismuth slimes, bismuth is precipitated with sodium carbonate, dissolved in HCl and precipitated as oxy-chloride, washed, dissolved again as chloride and added to the bismuth baths according to requirements.

Notes on Electrochemistry and Metallurgy in Great Britain.

(*From Our Special Correspondent.*)

THE IRON AND STEEL INSTITUTE.

SECOND DAY'S MEETING.

Discussion on Steel from High Silicon Phosphoric Iron.—Two papers by Mr. A. Windsor Richards were down for joint discussion, but the author's appeal to the chair for separate discussion on account of their different characters was acceded. After a brief resumé of the first paper on the "Manufacture of Steel from High Silicon Phosphoric Iron" (abstracted on page 262 of our last issue), Prof. Turner opened the discussion by remarking on the wide interest of the paper, which revealed the fact that apparently manganese was no longer an essential constituent. The use of oxide of iron was the chief feature of the process. He believed that tentative experiments had previously been made along similar lines in South Staffordshire with some measure of success, but nothing definite had been done.

Mr. Arthur Cooper, speaking of experiments made at the North Eastern Steel Works along the same lines which had not been so successful, said, in Mr. Windsor Richards' works they used larger converters and also better ore. What with delays and so forth, the net result was that whilst they produced a fairly good steel it was not so good as steel made from ordinary basic iron containing manganese, and was appreciably dearer in cost. As to phosphorus, it was just as easy to make good steel from pig containing 2½ as 1½ per cent phosphorus, so long as the pig was uniform.

Mr. Vaughan Hughes asked for analyses and tests of the steel produced by this process.

Mr. P. C. Gilchrist stated that the paper was not only exceedingly interesting but was theoretically correct. He thought that basic steel of the future would be made in the open-hearth furnace and not in the Bessemer converter, and that Mr. Richards' process would have its most successful applications in the open hearth.

Mr. Saniter expressed his pleasure at the author's concurrence with views which he had long held that manganese was unnecessary in steel marking. This point was replied to by

Mr. Riley's remark that manganese was not necessary with pure iron or steel, but merely a physic for impure irons. Mr. Paul said that the paper before them dealt with a hard rail steel. Did the same conclusions apply to structural steel? Mr. Twynam asked for complete analyses of the slag, and asked whether the lime came from the sides of the converter.

Mr. A. Windsor Richards, in reply, said that their experience certainly differed from Mr. Cooper's. It was expensive to put manganese and phosphorus into iron, and more expensive to get them out. He promised to give Mr. Hughes the information asked for, and disagreed with Mr. Gilchrist's view as to the open-hearth furnace, and stated that he considered that the Bessemer process had now a long lease of life.

Discussion on Steel from Pig Iron Containing Chromium, Nickel and Cobalt.—Mr. Richards' second paper (abstracted on page 262 of our last issue) was briefly introduced. Mr. Saniter opened the discussion and characterized the subject as both novel and interesting. Further, the ore used would be really valuable if the process could be worked economically. The removal of the second and third lots of slag would be more difficult than the sulphur; perhaps a tilting furnace would be the outcome. As to the author's test figures further particulars would be of value giving the elastic limit and tensile strength.

Mr. Vaughan Hughes said that, in experimenting with armor piercing shells, he had produced a similar steel with more carbon.

Mr. Henderson enquired whether a pig iron with 0.5 per cent of chromium would permit of fairly easy working in an open type of furnace. Unless this were possible the ore in question would be used only as a specialty.

Mr. Gilchrist asked for a sketch of the furnace, and remarked that the new steel was very beautiful and would have wide uses.

Mr. Cooper, in a reminiscent vein, told how thirty years before, in Sheffield, they had received a pig iron from Tasmania with 6.2 per cent of chromium. They made up a charge with one-fourth of this, and rolled into rails and obtained results which so astonished them that they reserved this stock of pig to help them out on special occasions.

Mr. Riley mentioned that 2,000 tons of the said pig iron, which was a white iron with 0.2 per cent of sulphur, came to this country. He had tried to puddle it and entirely failed, and Mr. Cooper had purchased the entire quantity at a very low price.

Mr. Stromeyer narrated a case of brittleness in chrome steel which bent perfectly well immediately after rolling, but was quite brittle in twelve weeks, and suggested that while chromium might initially improve steel it might cause depreciation later on.

Mr. Lester asked for complete analyses of the final slag, and anticipated the time when ores containing chromium such as existed in large quantities in India would have to be worked.

Mr. Duffield said that he had attended to the making of 8,000,000 tons of Bessemer steel, and had used chrome iron to help him out of difficulties, and had had some of his best steel when he had added pig containing from 3 to 4 per cent of chromium.

In reply, Mr. Richards spoke chiefly upon the question of slag removal, the chief point regarding which was to remove the slag before it got too thick. As regards the order of the elimination of the metalloids it was very simple. The chromium oxidized rapidly with the silicon, and after the silicon was removed only the carbon had to be got rid of.

Steam in Gas Producer Practice.—The interim paper by Dr. W. A. Bone and R. V. Wheeler (abstracted on page 263 of our last issue) provoked a long but not very interesting discussion. The speeches were over-long, and Sir Hugh Bell had to hint twice to one speaker that he should be brief. After the second rebuke from the presidential chair, the audience closured the speaker, discouraging further remarks by mild but continued stamping. If more audiences would so closure verbose orations it would add to the respect in which scientific meetings are held. In the discussion, Mr. Armatage spoke about high temperatures as giving best ammonia recovery, high thermal efficiency and freedom from clinkering. Mr. Sahlen remarked that producers were too frequently left to take care of themselves, whereas they required intelligent management. Mr. Thwaite urged the superiority of carbon monoxide in comparison with hydrogen, and pointed out that the proportion of carbon monoxide increased with the height of the incandescent fuel in the producer. He further expressed the view that steamless air-blown generators were more efficient than steam ones. Mr. Lynn, in a very lengthy speech, transgressed the ten minutes' rule, lost the attention and sympathy of the meeting, and incurred the rebuke of the chair. The speaker's chief point was that the author's figures of efficiency would not apply to a Mond plant working under normal conditions. Mr. Stobie favored gas producers into which steam was not injected, as in making steel bars for Sheffield crucible steel makers, no oxidation took place with open-grate producers; injected steam led to a great oxidation of the metal, and after all water was not a fuel. Mr. Walton Dixon said that the ultimate practical effect of the paper became, whether steel makers should go in for ammonia recovery or not. The authors presented the alternative that if gas was wanted ammonia need not be reckoned upon; if plenty of ammonia was wanted, people need not trouble about the gas. Commercially and theoretically they might be correct—from the practical standpoint they were not correct. Actually if gas was needed for reheating furnaces the ammonia must be reduced. Prof. Bauerman asked for the ratio of air to water in the blast used. As regards the Mond producer it should be worked in its entirety.

In reply, Prof. Bone hit back trenchantly at his critics, as he and his co-author had directly disclaimed any attempt to deal with the efficiency of the Mond process, and that his critics had gone beyond the limits of the paper.

The Effect of Manufacture on the Properties of Steel.—Unless the meeting was completely prostrated by the long and not wholly relevant discussion of the preceding paper it is almost impossible to account for the absurdly brief consideration given to Mr. F. W. Harbord's valuable paper (abstracted on page 263 of our last issue). This was as compact and impartial a review of the subject as could possibly be conceived. It may have been that its comprehensive survey was so just that the friends of different methods of manufacture were left without any case whatever to present. There were only two contributions to the discussion: the first by Mr. P. C. Gilchrist, who asked Mr. Harbord for an additional column giving the amount of oxide of iron present, as he was convinced that the difference in hardness between the four classes of steel was due to this factor. Then Prof. Turner regretted that the author had merely contented himself with recording facts, and had advanced no theories to account for the differences in hardness. In reply, Mr. Harbord promised to endeavor to carry out Mr. Gilchrist's suggestion, for which he thanked him. As regards Prof. Turner's enquiry, he had thought it was wiser to define the facts rather than advance explanations.

Ageing of Mild Steel.—The meeting's reception of Mr. C. E. Stromeyer's paper (abstracted on page 264 of our last issue) was, on the whole, suggestive of toleration extended to a brother weak in the faith, rather than the denunciation of a creator of bogies which might have been expected. Perhaps, because the author had too large an assortment of horrible examples of fractured steel plates to throw (metaphorically, of course,) at their heads.

Mr. Saniter suggested that one source of boiler failures lay in the sudden cooling to which some boilers were subjected at the time of cleaning—brittleness due to shortening following. While he agreed with the author that some steels did show a deterioration on ageing, other steels showed a distinct im-

provement. Those steels most subject to ageing contained oxide of iron.

Mr. Ainsworth commented on the fewness of boiler explosions and accidents to structural material, and urged that Mr. Stromeyer's cases must be regarded as quite exceptional.

Mr. W. Rosenhain urged that Mr. Stromeyer's cases were due to severe local strains, and proposed a long systematic series of tests as to the ageing of steels of normal and severely strained composition.

Mr. Duffield talked about the dangers of piping in soft ingots, and depreciated the use of too soft steels. They wanted more carbon and metal free from oxygen.

Mr. David Colville expressed his view that ageing did not take place. Strains might have been occasioned by the rolls being slightly out of truth, such strains being locked up in the plate. Mr. Stromeyer ought to look for these and not for ageing. The temperature and time occupied in annealing were important.

Mr. Hall referred to the condition of the charcoal iron used in the dismantled Niagara Bridge. After thirty or forty years it was found that the strength of the plates was within 5 per cent of the initial value.

In reply, Mr. Stromeyer admitted the possibility of local strains, and deprecated repeated hydraulic tests year by year. A calking effect was the same as nicking and set up strains. He did not think Mr. Rosenhain's series of experiments necessary. Discussing the relation of ageing and fatigue, Mr. Stromeyer expressed the view that these actions were converse, and that material which would fatigue easily would not age easily, while materials which aged rapidly would not fatigue easily.

Sir Hugh Bell's vote of thanks to Mr. Stromeyer, the reading without discussion of a thermo-dynamically weak paper by Mr. A. J. Capron on "Induced Draught for Steam Raising," and a graceful reference from the chair to the undiscussed papers, ended the most business-like session which the Institution has held for many a long day.

Exhibits at the Royal Society's Conversazione.

Apart from gyroscopes—for tight-rope trains on the Brennan system or on the Schlick system to reduce the rolling of vessels—there was very little that was epoch-making in the exhibition on May 8. Three features, however, call for mention.

The Cambridge Scientific Instrument Co. exhibited Prof. Féry's self-contained radiation pyrometer. This pyrometer utilizes the heating effect of the "total radiation" from a hot body, focused as an image of that body by means of a concave mirror. It differs from the already known Féry radiation pyrometer in being entirely self-contained, the image falling on a minute bi-metallic flat spring (Breguet spiral). This becomes partially uncoiled as its temperature rises, and a light pointer attached moves over a dial divided to give direct temperature readings.

The same company also exhibited the universal portable electrometer, designed by C. T. R. Wilson, F. R. S. This is a gold-leaf instrument, with very small capacity and fused quartz insulation. It is suitable for work on atmospheric electricity, radio-activity, etc., and is self-contained, with a means for standardizing the readings, which are very steady even in a fairly high wind.

Dr. F. D. Chattaway, F. R. S., had an interesting exhibit of copper mirrors, produced by depositing copper on glass from an aqueous solution. In the mirrors exhibited the copper had been deposited upon the glass by reducing cupric oxide by an aqueous solution of phenyl hydrazine in presence of potassium hydroxide, which accelerates the action to a remarkable extent. The mirrors are equal in brilliancy to silver mirrors, and on account of the color of the copper are much more beautiful.

The Faraday Society.

My notes on the discussion at the Faraday Society's meeting, held on June 25, must be held over until next month. The evening was devoted to a general discussion on "Hydrates in Solution." As texts or expositions of rival theories four papers were presented, the authors being Messrs. Bousfield & Lowry, Mr. J. C. Philip, Dr. Fundley and Dr. G. Senter. A large audience was present.

The Engineering Conference.

The Fourth Conference convened in London in June by the Institution of Civil Engineers covered a most varied field of subjects. It was, of course, entirely sectionalized, but while there was a distinctly metallurgical section devoted to mining and metallurgy, metallurgical papers were also to be found among such sections as "railways," "ship building" and "applications of electricity."

Postponing all mention until next month of metallurgical papers read in other than the railway section, mention must first be made of Mr. C. P. Sandberg's paper on "The Chemical Composition of Steel Rails and Latest Developments." The author, after declaring that a universal specification to suit all cases could not be arrived at, discussed the effect of different elements. He thought engineers had suffered enough from fractures due to a high percentage of phosphorus, especially in cold climates, to make it necessary to limit it as far as possible, in spite of the good wearing results of high-phosphorus rails.

Sulphur probably presented more trouble to the manufacturer than to the user, because if a high percentage is present the rails will be red-short. No more manganese should be allowed than is absolutely necessary for clean rolling, as he had found many cases of rail fractures attributable chiefly to high manganese.

He had for many years experimented with silicon in rails with a view to obtaining soundness and solidity without brittleness, and had found that for rail steel the effect of silicon added is very different from the effect of silicon left in from the pig iron. When silicon is left in the percentage varies considerably, depending on the heat of the charge, thus causing great irregularity, and as the iron has not been completely converted into steel the metal is of a brittle character. But when the silicon from the pig iron had been eliminated as far as possible and a known quantity of silicon added, in the form of high-percentage silico-spiegel or ferro-silicon, he obtained regularity in the percentage of silicon, and, moreover, the silicon then toughened the steel instead of making it brittle, this being largely due to the more complete removal of gases and oxide from the steel.

In general it is desired to obtain a rail as hard as is compatible with safety, and carbon is the most suitable hardener. The author did not specify carbon limits, merely stating that it shall be as high as the safety or drop test will allow. Although not absolutely a criterion the drop test is still the best safety test we have.

Considerable trouble had been experienced on electrified railways, owing to the excessive side wear of the rails on curves. Apart from the question of better wearing steel for their rails in general, one of these railways has introduced the novelty of using a considerably harder steel for check rails. This had shown surprisingly good results, about which he hoped more would be heard during the discussion.

The general purport of the discussion was the consideration of the value of silicon as an ingredient in rail steel. Mr. Ross, of the Great Northern Railway, commenced with a statement of his experiences. He said that it was by no means easy to lay down any fixed rule for the composition of rails. The inter-relation of the constituents were complicated and uncertain. He had some Sandberg rails down for twelve months, and found them very good. They contained .441 per cent silicon, .264 sulphur and very little phosphorus. They had tried a few rails with 3 per cent nickel in them; they were most excellent but far too dear for general use. He had great hopes for the silicon rail.

Mr. Willcocks, of the Metropolitan, maintained that the rails

made twenty-five years ago were much better than the present rail. He considered, at considerable length, the abnormal wear which took place with electric traction. Rails with steam traction lasted in a tunnel five years; outside twelve years. With electricity the time became two years inside and five years outside. He then gave a number of figures, all going to show that the Sandberg rail was much more durable than any other. The wear on the Metropolitan was peculiar in that it was largely side wear. The question was to what this side wear was due. He held that the system of traction was to blame. The train was partly pushed, partly propelled, and the pushing tended to set the bogie against the sides of the rails. Old rails were used as check rails on curves. They lasted about 194 days, losing as much as 7.7 pounds per yard per month. Sandberg hard rails lasted for four months without much visible signs of wear.

The most interesting contribution to the discussion was made by Mr. Short, in charge of the London & South-Western Railway laboratories, who gave particulars of an investigation of 100 broken rails fractured within the last two years. He found that phosphorus up to 0.14 per cent had little or no effect, rails with that amount having lasted twenty-four years. Not one fracture could be traced to that element, which, by giving hardness, prolonged the life of a rail. Sulphur up to 0.145 per cent, silicon up to .2 per cent, and manganese up to 1.5 per cent had no effect. As to carbon, they found percentages of .3 and .52. The latter rail lasted three times as long as the former. With .45 per cent the fracture took place through the fish-plate bolt holes; with .4 per cent they broke clean. The fractures were invariably near the end of the rails. Of the 100 fractures thirty were obviously due to slag rolled in, 60 per cent occurred through solid metal. Of 200 tons of rails used for experiment the wear was found to average 1 pound per yard per year. He held that the modern rail lasts twice as long as the old one.

Mr. J. E. Stead said that the great merit of silicon was that it promoted soundness. Piping could be diminished by teeming very slowly. Much of the differences in results depended upon whether the rails were rolled in summer or winter. The rapid cooling of the rail made it brittle, particularly if there was much manganese present. Phosphorus had a tendency to segregate in patches, and the rail to break through a patch; silicon reduced segregation.

Omitting any references to a paper on reinforced concrete I must mention a brief paper by Mr. Bertram Blount on "The Best Means of Preserving Iron and Steel Work in Railway Construction," which expressed the opinion that the best preservative for iron and steel work exposed to the weather is some bituminous preparation. Natural bitumen is very permanent, but rather costly; coal tar, properly boiled so that the coating shall be neither brittle nor sticky, is almost as durable and is cheap enough. Where the work is exposed to boiler gases, *e. g.*, overhead bridges and station roofs, complete coating with a good layer of hot bitumen has so much practical advantage over paint that no question of appearance should be permitted consideration. For railway structural steel work, such as the frames of buildings or frame foundations, there was, he considered, no preservative so good as lime or cement concrete. This applied whether the steel framing was bedded in concrete or whether it formed an integral part of the concrete, as in the various "ferro-concrete" systems. It was essential that the concrete should be dense and rich in cement, and that the cement or lime should be unexceptionable. Steel protected in this way need not first be cleaned from scale and rust.

The resultant discussion was in the main favorable to Mr. Blount's view, one speaker alone stating in dissent that tar had done much mischief owing to its acid contents.

Mr. R. A. Mallock's paper on "The Action Between Rail and Wheel" was purely physical in its contents, and, assuming an absolutely rigid rail bed, the author tentatively put forward some hypothetical formulæ, which critics in the audience did not accept.

Market Prices During June.

In the chemical trade the price of copper sulphate has receded during the month from £33.10 per ton to £32.10, and ammonia sulphate risen 2s. 6d. to £12.10 per ton. Montreal potash is unchanged. Bleaching powder is quoted at £4.12.6 per ton, and white caustic soda, 77 per cent. at £10.12.6 per ton. Zinc sulphate is quoted at £8.

In the metal market, copper has been in a very unstable condition. Opening at £101 per ton the price of £95 was touched on the 13th, rallying to £101; on the 18th and 19th prices broke again, falling to £94 on June 24, and finally closed at £98 on June 29th. Tin prices have also been unsettled, declining to £185 on June 10 and rallying to £191 on June 18, receding to £188 on June 21, and returning to £191 on June 25, and closed at £192.10. Lead opened at £20.17.6 per ton, and commencing to rise on June 7 reached £22.10 on June 17, and commenced to fall three days later. The final price was £21.10.

The chief feature of the month has been a weakening of the iron market, Cleveland pig iron breaking from its high price of 61s. 10d. and falling by the 14th to 55s. 10d., remaining in this neighborhood for about ten days, and closing at 56s. 3d. Hematite prices also broke away from their opening value of 79s. 10d. per ton to 75s. 6d. by the 13th, touching 76s. 10d. on the 19th, with subsequent irregularities, and then rising to 81s. 9d. Antimony has closed at £50 to £55 per ton, thus relieving a strained situation, during which prices once touched £115. Platinum is quoted at £5 per ounce.

London, July 6, 1907.

SYNOPSIS OF PERIODICAL LITERATURE

A Summary of Articles Appearing in American and Foreign Periodicals.

Electric Smelting of Iron Ore in California.

Heroult Process.—We have already referred repeatedly to the work now going on in Shasta County, California, with a view to electrically smelting iron from the large magnetite deposits there available. The *Mining and Scientific Press* of July 20 contains considerable information on the progress of the work both in an editorial and in an article by R. L. Phelps. The deposit of iron ore consists of a large body of magnetite, uniform in texture and analysis, containing from 68 to 70 per cent iron, that is, only 2½ to 4 per cent of impurity. (An analysis gave 70.2 per cent Fe, 2.4 SiO_2, 0.012 S, 0.01 P and 2.4 insoluble.) Although as yet only partially explored, there is evidence to warrant the expectation of a plentiful supply of ore and it is situated as to be conveniently transported to the smelter. The magnetite appears to be associated with diorite and it lies at the contact with a limestone so pure as to serve as a flux in smelting. The iron mine and the limestone quarry are alongside. It is expected that the ore can be delivered for $1.50 per ton to the smelter. Electric energy is obtained from the Northern California Power Co. at $12.00 per horsepower-year in the form of three-phase currents. There was some fear that trouble might be experienced with the three-phase electric furnace by short-circuits between the three electrodes so that the current would not pass through the neutral pole at the bottom of the furnace, which is made of a carbon paste tamped tight and of the same material as the electrodes themselves. However, such fears proved groundless. The experiment is made under favorable commercial conditions, since the best pig iron sells in San Francisco at from $30.00 to $32.00 per ton. It comes by sea from Europe, mainly as ballast, and pays a heavy duty. No iron

is made in California, mainly because a suitable cheap fuel is lacking. For smelting in the electric furnace, charcoal is used as reducing agent, and unlimited timber is available for this purpose at the smelter. It is estimated that the magnetite in Shasta County can be converted into pig iron and placed in San Francisco for $15.00 to $18.00 per ton. This experiment is being undertaken by Mr. H. H. Noble, president of the Northern California Power Co. It is made at his own personal expense. The engineer in charge is Dr. Paul Héroult, assisted by R. Turnbull, N. Petinot and E. Humbert. Three-phase currents, at 30,000 amps., 50 volts, 60 cycles are delivered to the electrodes from transformers supplied with the 22,000-volt current of the transmission line.

The smelter itself is elliptical in form. The bottom of the furnace is formed of heavy cast-iron plates with a covering of tamped carbon to form the neutral point. A tap and trough are provided on one side to draw the molten pig iron onto the molding beds. Owing to the small amount of slag produced it is not necessary to provide a tap for removal of the slag. The ends of the electrodes are kept in the slag rather than in the molten metal. The carbon electrodes are held in copper holders which are water-jacketed. A diagram of the furnace is shown in Fig. 1, where A is the chamber leading

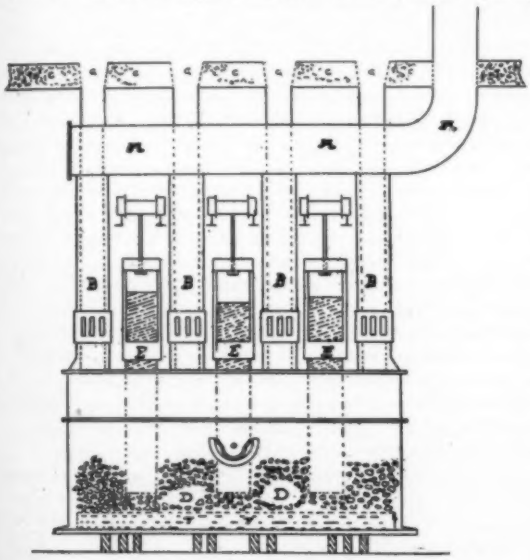


FIG. 1.—HEROULT THREE-PHASE FURNACE FOR SMELTING IRON ORE.

gases from the furnace to the stack; B are combination charge and draught tubes; D is the charge in the furnace and E are the electrode holders. The charge is made of charcoal, limestone and ore. The charcoal is burned in kilns close to the plant.

Water for cooling the water-jacketed transformers and electrode holders is furnished by an 8-hp. centrifugal pump. The current was turned on for the first time on June 29 to dry the lining and test the electrodes. On the evening of July 3 the current was again turned on and the furnace warmed up for the formal start on the morrow. "About 10 o'clock on July 4, the furnace being sufficiently heated, a charge was added and the real experiment began. The daughter of Mr. H. H. Noble had the honor of feeding the first shovel-full of ore. With a low steady hum of the current the smelter ran successfully for 3½ hours, reducing the charge until molten iron collected in the bottom. This proved that the furnace could be operated by a three-phase 60-cycle alternating current. At the end of 3½ hours difficulty was found with the centrifugal pump." Another trouble was later experienced on account of clogging of a charging tube and just as this was being corrected the transmission line was broken by an accident and the supply of energy stopped. However, the repairs have been made and the editorial concludes with the following statements: "As we go to press we learn by telephone from the smelter (280 miles from San Francisco) that five tons of iron have been tapped, that the quality of it is excellent, and that successful results are confidently anticipated."

ELECTRIC DISCHARGES THROUGH GASES.

Ozone.—A Franklin Institute paper by J. H. Bridge, published in the May issue of the *Journal* of the *Franklin Institute* deals with ozone, its nature, production and uses. (An ozonizer of the author has been in use side by side with the Vosmaer ozonizer at the experimental plant of the United Water Improvement Co. of Philadelphia.) The author reviews the different forms of electric discharges through air and points out that the one discharge which produces ozone is the silent discharge which is characterized by a dark blue violet color, being a convective discharge, unidirectional from positive to negative. The theory has recently been advanced, and is now gaining pretty general acceptance, that the formation of ozone by the brush or silent discharge, is caused, not by any direct electrical action on the oxygen itself, but is simply a photo-chemical effect, due to the ultra-violet rays accompanying such discharges. This theory originated with the discovery of Lenard, who noticed that if the light from a spark gap was allowed to pass through quartz plate, which is transparent to ultra-violet rays, and to impinge on oxygen, ozone was generated. If, however, a substance impervious to ultra-violet light was interposed, no such action took place. In most forms of ozonizers high-tension discharges are employed, a dielectrics—like glass, mica, or vulcanite being placed across the path of the current to prevent the formation of sparks and arcs. The trouble is, however, that these dielectrics often break. (Although it is not mentioned in the paper it may be stated here that the Vosmaer system does not employ any dielectrics but prevents the formation of arcs and sparks by purely electrical means. The recent failure of the United Water Improvement Co. in Philadelphia is believed to be due to the fact that while the troubles of a dielectric are avoided in the Vosmaer system, yet new troubles were introduced by the special condensers which are required in the system and which are reported to have not proven so fully reliable as is required for commercial work. In the Bridge system, which was also used at the Philadelphia plant, a dielectrics is used but not of glass but of a stronger and a more resistant material so that it is not as liable to break.)

As regards the various yields of existing types of ozonizers, the differences in methods of testing render comparisons unreliable; and, until a uniform method is applied to determine the ozone—content of a given body of gas, no exact comparison can be made between the efficiency of the different types of apparatus now in use. The Siemens & Halske ozonizer, with dielectrics, is reported to yield 20 to 30 grams per kw. hour. Elworthy claims 60 to 70 grams from a machine of similar construction; but no data are available as to the richness in ozone of this product. Otto has published figures showing yields of 20 to 118 grams per kw. hour, the former where the concentration was .852 grams per cubic meter of air, and the latter where it was only .542. Vosmaer with a concentration of 1.5 obtained a yield of 12 to 16 grams. The de Fries apparatus produces about 20 grams with a concentration of 1.0. "Generally speaking, 20 grams per kw-hour has hitherto been considered a fair output if the air treated contains one gram and upwards of ozone per cubic meter." Bridge states that he has improved the efficiency by making use of the fact that the silent discharges are always in the form of hollow cones of light or of cone-shape funnels. In all ozonizers hitherto devised, the air is directed against, or around, these cones of light in a plane at right angles to them. Now it is a well known fact that when electricity is discharged from a

point, the surrounding particles of air are electrified, and, being of the same electrical sign, repulsion takes place. The currents of air repelled from the points of a discharging electric machine may be felt by the hand, or seen in the deflected flame of a candle held close to them. With these facts in mind, it is obvious that air, when passed into the influence of a discharge at right angles to its path, is immediately thrust away from it, and any effect which the discharge may have

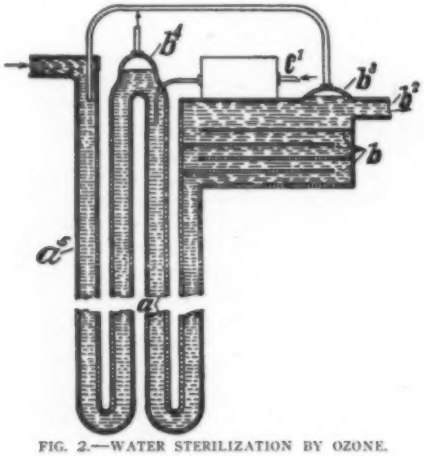


FIG. 2.—WATER STERILIZATION BY OZONE.

on it is but momentary. To overcome these disadvantages Bridge has devised a perforated electrode, and has so "arranged the air supply that it passes through the perforations in fine streams directly into the hollow cores of the discharges taking place on its surface. In this way the air is forced to travel upwards with the discharge, while completely surrounded by it, and then forced through the luminous walls of the discharge. This brings every particle of the air into intimate contact with the discharge. The first tests of the apparatus gave 80 grams of ozone of a high concentration."

The second part of the paper deals with the applications of ozone. Ozone is useful in various chemical manufactures and it is stated with respect to the manufacture of vanillin that in a single month in 1903, the Société Française de l'Industrie Chimique produced 22,000 kilograms of vanillin by means of ozone, which at 35 to 45 francs has taken the place of a product that in 1895 was worth 800 francs. (The same process is also used at Niagara Falls.) Ozone is also useful for bleaching and purifying purposes as well as in thereaputics. The chief application is, however, in the sterilization of water. (A great many sterilizing apparatus have been described and illustrated in former volumes of this journal.) Fig. 2 shows the Howard-Bridge system in which the suction action of the water under treatment draws into itself the ozonized air required for its purification. The excess of ozone which remains unabsorbed and which in some systems is run to waste is here utilized in the preliminary treatment of the water. Raw water enters the pipe a^2 drawing, by suction, unabsorbed ozone from b^3. The waste gases escape at b^4. The current of water in a sucks fresh ozone into itself from the ozonizer c; and after passing around the baffle plate in b, the prified wates escapes at b^2.

Precious Metals.

Advance of Cyanidation.—In our Vol. IV., page 345, we had an interesting article by J. W. Richards on the metallurgical revolution in Guanajuato, Mexico, due to the introduction of the cyanide process. The *Mining and Scientific Press* of June 1 states that Guanajuato and Pachuca are seeing rapid advances in cyanidation and the passing of the patio process. At Pachuca excellent results have been obtained by A. Grothe in his new plant, consisting of five conical-bottom vats, 40 feet high and 15 feet diameter, in which sand and slime are agitated by air. Six horsepower suffices to agitate 100 tons of ore. The extraction is rapid. The cyanide plant of the Hacienda San Francisco has been in operation six weeks, and so successfully that plans are being drawn for changing several pan-mills to cyanide plants. The owners of the San Rafael mine are about to add twenty stamps to the forty already at work, using three tube mills for regrinding. The pulp will be crushed in solution and will then be agitated by compressed air in both of the vats. Butters filters are to be employed for treating the residue. At Guanajuato there is similar activity. A 1,000-ton plant is assured for the La Luz mines. The plans call for 320 stamps, concentrators, regrinding machinery and a cyanide annex. Machinery for a 60-stamp mill and cyanide plant is reported to have been ordered for the Tajo de Dolores mine. The *Mining and Scientific Press* of June 29 contains an illustrated description of old and new methods at Guanajuato from the facile pen of T. A. Rickard.

Homestake Mills.—In *Mining and Scientific Press* of July 6, F. L. Bosqui gives an account of present cyanide practice at the Homestake Mills in South Dakota. There are six stamp mills containing an aggregate of 1,000 stamps and crushing about 4,000 tons per day. The most interesting feature of the mill practice is the amalgamation. Each battery is provided with four full-size plates, in series, each plate being 54 x 144 inches and ⅛ inch thick. The first is plain copper, the last three are silver-plated. This addition of three silver plates, giving to each 10 stamps of the Amicus (240-head) mill, a total plate area of 600 square feet, and to the other mills an average of 360 square feet per 10 stamps, is a comparatively recent innovation, and has proved an excellent one, increasing the recovery by amalgamation approximately $200,000 per year above the annual recovery when only one 12-foot plate was used. The saving by amalgamation is between 70 and 75 per cent. The leachable portion of the tailings is treated after slime separation at two cyanide plants. The most interesting feature is the treatment and filtration by means of filter processes as devised by C. W. Merrill. At the more recent of the two plants the total cost of treatment per ton during the last six months of 1906 was between 16.1 and 18 cents.

Slimes Filter.—In our March issue, page 88, we published a description of the Butters slimes filter. A very elaborate article on the filtration of slime by the Butters method is given in the *Mining and Scientific Press* of June 22 and 29 by Mr. E. M. Hamilton.

Ore Dressing.

Acid Flotation Processes.—In our Vol. IV., p. 49, we gave a review of acid flotation processes. In the *Mining and Scientific Press* of June 8, F. H. Jackson gives an illustrated account of the practice of the acid flotation process at Broken Hill. The object is to separate the blende from the gangue. The tailings from the lead concentrating mills consist of a mixed product of blende, garnet, quartz, rhodonite and calcite. Since the blende has practically the same specific gravity as rhodonite and garnet, no method based on gravity can be used. The Potter and Delprat patents are both for "acid notation" processes and both depend on the power of an acid solution to float the particles of zinc blende on its surface, from which they may be mechanically and continuously drawn off. Both processes are practically the same, and the same type of plant required for each, but while a weak acid solution is all that is required in Potter's method, the Delprat process requires the use of salt cake, which, it is claimed, increases the specific gravity of the solution, and therefore its floating powers. This salt cake is acid sodium sulphate ($NaHSO_4$), and is a by-product in the manufacture of sulphuric acid at the Proprietary Co.'s works. Whether it is of any material benefit is doubtful. The strength of the acid solution used depends on its temperature and on the alkalinity and freshness of the tailings treated; it varies from 0.5 to 1.5 per cent H_2CO_4. Very old tailings and those which have become oxidized, consume more acid than would be required to treat them

as they came from the lead concentrators in dissolving the incrustations from the faces of the blende particles before concentration by flotation can take place. The temperature of the solution is a most important factor, it should be kept as high as possible, and not lower than 80° C. Testing of solution in the pans for temperature and acidity should be frequent throughout the shift, the latter being tested by titration with normal sodium hydrate, using methyl orange as an indicator. Material between a 40-mesh and an 80-mesh screen was found to be the best in size for treatment, and old tailings should be sifted through coarse screens to extract stones and other rubbish, which cause trouble in the pans. What actually happens in the pans is a little obscure, but the following is an approximation: "The calcite and other carbonate particles in the tailing, together with the iron in the blende, are at once attacked by the acid, and bubbles of CO_2 and H_2S are respectively formed. For some reason the bubbles of CO_2 leave the calcite, etc., and attach themselves to the bubbles of H_2S surrounding the particles of blende, thereby causing these to float to the surface of the liquor; the flow of liquor carries the blende and bubbles into boxes, where, on the bursting of the bubbles, the blende particles sink and are drawn off. The quartz, rhodonite, etc., in the gangue sink to the bottom of the pan without hindrance and are tapped, the process being continuous." An illustrated description is then given of the practice in a Broken Hill mill using a powder process.

MISCELLANEOUS.

Coal and Oil Muffle Furnaces.—The *Mining and Scientific Press* of June 1 contains an article by G. J. Young, in which a coal and an oil muffle furnace were compared, which were identical in all dimensions with the exception of the two combustion chambers, the combustion chamber of the oil furnace being altered to admit of better flame distribution than could be obtained by using the coal combustion chambers. The main advantages of the oil furnace over the coal furnace, apart from the slight one of operating costs, are its cleanness, ease of regulation, the high temperature quickly reached and easily maintained and the moderate amount of attention required. The principal objection is the noise of the burners. Temperature curves given in the article show the difference between the combustion of the coal and and oil. With the coal furnace, combustion takes place throughout the combustion chamber, with the oil the combustion takes place principally in the lower part of the chamber. This accounts for the small difference of temperature (15° C.) between the upper and lower muffles in the coal-fired furnace and the greater difference (140° C.) in the case of the oil furnace. The relatively lower flue temperature of the oil furnace is due to the same cause.

Melting Points.—An American Physical Society paper by J. K. Clement, published in abstract in the June issue of the *Physical Review*, gives the results of some new measurements with the gas thermometer, a conscientious effort having been made to reduce the magnitude of the errors common to the well known European gas thermometers. The following results of melting point determinations are given:

Zinc, 418.4°, greatest deviation from the mean, ± 0.4°.
Silver, 957.5°, greatest deviation from the mean.
Gold, 1,059.1°, greatest deviation from the mean.
Copper, 1,080.2°, greatest deviation from the mean, ± 0.5°.

The Detinning Suit.

Under the title "The Outcome of the Detinning Suit" we reported in our February issue, 1906 (Vol. IV., page 46), on the decision rendered by Vice-Chancellor Bergen, of the Court of Chancery of New Jersey, in the suit between the Vulcan Detinning Co. as complainant and the American Can Co. as defendant. This decision, which was in favor of the American Can Co., has now been reversed by the New Jersey Court of Errors and Appeals in a decision delivered by Judge Garrison on July 2.

On account of limitations of space we are unable to report at length on this decision, which is exceedingly interesting and important as a precedent with respect to the question to what extent so-called "secret processes" are protected. A full report is reserved for our next issue.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

ELECTRIC FURNACES.

Induction Furnace.—E. A. Colby, 859,641, July 9, 1907. Application filed Nov. 23, 1905. Assigned to American Electric Furnace Co.

A vertical cross-section of the furnace is shown in Fig. 1 and a horizontal cross-section of part of it in Fig. 2. The annular crucible 1 with the chamber 2 for the charge rests on the base plate 3 of soapstone, and is provided with a jacket 4 with a wall of asbestos board and filled with magnesia for heat insulation. The magnetic circuit is represented by 6, 8, 9, 13, 14 and 18. A special construction is used to secure firm contact between the core 18 and the upper parts 13, 14, and at the same time to make it easy to detach these parts; 7 are the trunnions for tilting the furnace. Around the core 18 and within the annular ring 1, the primary winding 24 is provided, consisting of turns of copper tube through which water circu-

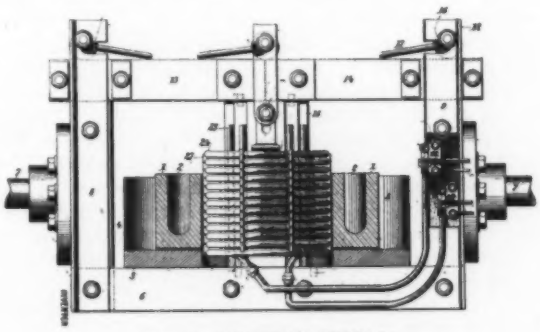


FIG. 1.—INDUCTION FURNACE.

lates for cooling. Interposed between the turns of the coil 24 are separating blocks 27 of refractory insulating material. As shown in the horizontal section, Fig. 2, the laminated core 18 is so constructed that vertical steep recesses are formed on its exterior. These recesses receive tubes 23 of porcelain fireclay, etc., which prevent contact of the core 18 and the primary core which surrounds it. There are 36 claims.

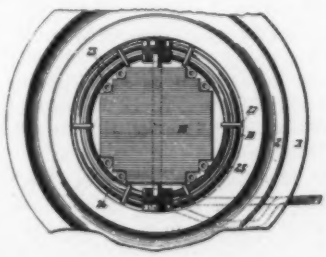


FIG. 2.—HORIZONTAL SECTION.

Production of Ferro-Silicon.—E. F. Price, 861,224, July 23, 1907. Application filed Aug. 31, 1904.

The furnace is provided with two electrodes suspended from the top and an arc is started between the electrodes and the carbon lining. A small amount of the charge is then fed into the bottom of the furnace, consisting of a mixture of silica, iron or iron ore, and coke, in which the ingredients are in such proportion as to make it a partial conductor when hot or which contains pieces of iron or magnetite and coke which tend to shunt the current. The furnace is then gradually filled until in its natural normal working condition the electrodes are

embedded in the charge. The minimum voltage is applied to the electrodes, to largely prevent the current from shunting through the charge. A layer of molten ferro-silcon collects in the bottom of the furnace. The high layer of charge surrounding the electrodes retains the heat within the furnace and protects the electrodes from the oxidizing and cooling effect of the atmosphere.

Manufacture of Silico Spiegel.—E. F. Price, 861,225. July 23, 1907. Application filed Nov. 14, 1905.

The process for making silico spiegel is quite analogous to that described in the preceding abstract for ferro-silicon. In the present case the charge consists of compounds of manganese and silicon and iron or iron ore together with carbon.

Detachable Hearth of Electric Furnace.—E. F. Price, 855,476, 855,477, 855,478, 855,479 and 855,480, all of June 4, 1907. Applications filed Nov. 14, 1905. 855,476 and 855,477 are assigned to Union Carbide Co.

In all these different patents the principal feature is that the electric furnace is provided with a detachable hearth or crucible which, when filled with the desired product, is withdrawn and an empty crucible is substituted. Preferably a series or chain of receptacles is employed to receive the product, each receptacle serving in turn as the hearth or crucible of the furnace; 855,476, 855,477 refer to the production of calcium carbide, 855,478 and 855,479 to the smelting of refractory ores and the production of ferro-alloys.

Vanadium and Alloys.—Fred. M. Becket, 858,325, June 25, 1907. Application filed June 19, 1906. Assigned to Electro Metallurgical Co.

To reduce vanadium from oxide and produce vanadium or vanadium alloys low in carbon, the inventor employs silicon and carbon as reducing agent, preferably in the combination of SiC. A basic flux such as slime is used to combine with any silica which may be present in the ore or concentrate, and with that derived from the oxidation of silicon. High yields are obtained of a product low in carbon, since the portions of the metal first reduced do not come into contact either with free silicon or free carbon, and therefore do not absorb the same to any injurious extent. The silicide of carbon and the ore are used in a fairly fine state of sub-division. To make ferro-vanadium or nickel vanadium, iron or nickel or their oxides are, of course, added to the charge.

Vanadium from Sulphide.—Fred. M. Becket, 858,328, June 25, 1907. Application filed March 5, 1907. Assigned to Electro Metallurgical Co.

The process avoids roasting and produces commercially pure metal directly from the sulphide. Commercially pure vanadium is produced by smelting in an electric furnace a mixture of vanadium sulphide and silicon or an alloy of silicon, the silicon being preferably employed in substantially the proportions required to combine with the sulphur of the sulphide or ore, the equation assuming the production of silicon sulphide, SiS_2. Commercially pure alloys of vanadium are produced by smelting in an electric furnace a mixture of vanadium sulphide with a silicon alloy, such as ferro-silicon or nickel-silicon, the alloy being preferably used in approximately the proportions required to supply sufficient silicon to unite with the sulphur of the sulphide or sulphide ore.

Reduction of Vanadium, Molybdenum, Titanium and Tungsten from Oxides.—Fred. M. Becket, 858,329, June 25, 1907. Application filed April 12, 1907. Assigned to Electro Metallurgical Co.

The process involves reduction in two stages. In the first stage partial reduction, specifically from a higher to a lower state of oxidation, is effected by a non-metallic reagent, such as carbon, hydrogen, carbon monoxide or a hydrocarbon; in the second stage the oxide is further reduced to a metal by silicon carbide. The first stage is carried out in an ordinary furnace, producer gas and water gas being particularly advantageous for this purpose. The second reduction is carried out in an electric furnace. In the case of vanadium the equation of the first stage is

$$V_2O_5 + 2C = V_2O_3 + 2CO,$$

and the equation of the second stage

$$V_2O_3 + SiC = V_2 + SiO_2 + CO.$$

The advantage of this combination process is that relatively inexpensive reducing agents are reduced in the first stage and the final reduction is effected by a relatively small amount of silicon carbide.

Low Carbon Metals or Alloys.—Fred. M. Becket, 858,780, July 2, 1907. Application filed Jan. 30, 1906. Assigned to Electro Metallurgical Co.

When the electric furnace is used for the production of low-carbon metals or alloys, aluminium or silicon may be used as reducing agent, but, nevertheless, some carbon will get into the metal, chiefly from the electrodes. To minimize this contamination the inventor contemplates the use of electrodes of very small sectional area. Since they have to carry a high current it is necessary to cool them in order to protect them. For this purpose a system of water cooling is used, as described and illustrated on page 279 of our July issue.

Alloy for Iron and Steel.—Fred. M. Becket, 858,327, June 25, 1907. Application filed March 20, 1907. Assigned to Electro Metallurgical Co.

The alloy which is stated to be particularly adapted for the treatment of iron and steel contains mainly titanium, calcium, aluminium and iron. The following two examples of the composition of alloys of this kind which the inventor has produced are given:

	Per Cent.	*Per Cent.*
Titanium	50.67	26.24
Calcium	8.32	5.30
Aluminium	22.70	1.55
Iron	11.83	50.98
Silicon	4.20	11.78
Carbon	1.86	3.69

The alloy is produced by reduction with carbon of a mixture of the oxides in the electric furnace. As compared with ferro-titanium the alloy possesses a relatively low melting point and is more rapidly incorporated with molten iron or steel and distributed through same.

Reducing Finely Divided Iron Oxides.—H. W. Lash, 860,922, July 23, 1907. Application filed Dec. 28, 1906.

As already noticed in our July issue, page 279, the process of the inventor contemplates the reduction in an electric furnace of a mixture of finely comminuted oxide of iron, such as iron sand or scale, with finely divided cast or pig iron and coke. Suitable iron oxide for treatment by this process are the sand ores of St. Lawrence and New Zealand, and their association with titanium oxide does not interfere with the process. Sawdust or crushed bituminous coal is a suitable addition, as are fluxes of the ordinary kind, such as lime and fluorspar. The following three mixtures are given as satisfactory for smelting:

	Mixture No. 1. Lbs.	*Mixture No. 2.* Lbs.	*Mixture No. 3.* Lbs.
Ore	16.00	12.00	18.00
Cast iron	16.00	6.00	7.00
Carbon	2.00	2.00	2.00
Limestone	.50	.25	.25
Fluorspar	.50	.25	.25
Sawdust	.50	.50	.50
	35.50	21.00	28.00

Electric Furnace.—Aldus C. Higgins, 856,061, June 4, 1907. Application filed April 6, 1906.

The object is to provide an electric furnace of relatively large capacity and so constructed as to permit the product to be formed and subsequently cooled under very uniform temperature conditions. For this purpose all the portions of the charge are brought within the cooling influence of cooled walls so as to secure commercial uniformity of the product. A relatively long and narrow furnace chamber is used in which are placed two rows of electrodes of opposite polarity at regular intervals. Between the two rows central interior walls are provided which are also cooled.

Electric Furnace for Pyritic Smelting.—H. Arden, 860,512, July 16, 1907. Application filed Sept. 24, 1906.

See the abstract under Pyritic Smelting in "Recent Metallurgical Patents."

Electric Zinc Furnace.—Fred. T. Snyder, 859,132, 859,133, 859,134, 859,135, 859,136, 859,137, all of July 2, 1907. Applications filed June 11, 1906, June 18, 1906, June 30, 1906, and July 25, 1906. Assigned to Electric Metals Co.

The principle which underlies all these patents is that in reducing zinc ore and condensing the zinc, liquid zinc will be obtained from fairly pure zinc vapor, but zinc dust from a zinc vapor diluted with other gases. Patent 859,132 (for the process) and 859,133 (for the apparatus) relate to the use of an electric furnace shown in Fig. 3. For the treatment of zinc lead ore, the ore is first roasted down to about 8 per cent in sulphur and the furnace charge is made up of the roasted ore mixed with coke or charcoal and with fluxes, such as lime and iron in such proportions as to form a slag which

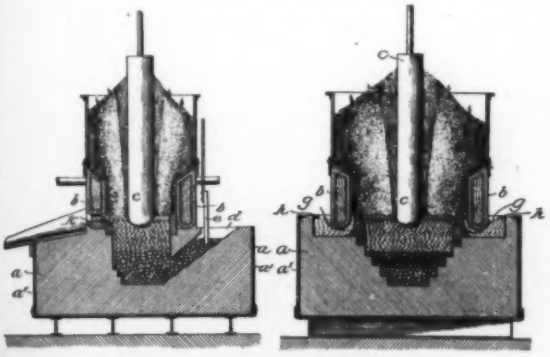


FIG. 3.—ELECTRIC ZINC FURNACE.

requires a high temperature for its formation. Such a slag will be high in lime and silica, the proportion of silica being about 50 per cent or more, and such a slag will not retain any appreciable quantity of zinc. The charge is fed into the rectangular stack furnace, being heaped up around the central electrode *c*, so that in general the coarser particles of the charge will roll toward the outside of the mass. Coke is also added around the outside of the electrode. The bottom of the crucible is filled with molten lead, forming the other electrode, connection being made to the outside to the conductor *e* dipping into the lead well *d*. A layer of slag rests on the molten lead and above the slag the charge of ore, etc., rests. The zone of maximum heat will be around the lower end of the electrode *c*, and the materials of the charge will there be reduced. The lead which is produced sinks to the bottom and adds to the body of the molten lead in the crucible, from which it may be ladled out from time to time through the lead well *d*. The slag which is formed is molten in the vicinity of the electrode, but around the hollow walls *b* which are water-cooled, the slag is congealed to form a lining for the walls. The CO gas which is formed escapes through the ore body to the top, where it may be burned. The conditions are such, however, that the escape of the gaseous product is restricted, so that a comparatively high pressure is maintained in a central portion of the ore body. Under such conditions the zinc is condensed in liquid form at the end walls *b*, and may be drained off under the walls through the wells *h*. The system of water-cooling the walls is so arranged that the greater volume of water flows through the end walls, so that there is a greater tendency for the zinc vapor to condense at the ends of the furnace than near the sides within the zone of maximum heat. The zinc vapor which attempts to escape with the CO gas upward becomes condensed in the ore body on account of its lower temperature and is condensed in the form of zinc dust. While the ore is progressively fed downward toward the heat zone, the zinc dust is again brought into the zone of maximum heat and revaporized, the result being that the mixture of gas in the furnace is progressively enriched until the percentage of zinc in the mixture is sufficiently high, so that the zinc condenses in liquid form in the wells *g*. The author states that, broadly, his invention involves heating the mixture to be smelted in successive stages; first to a temperature sufficient to reduce the compound, removing the non-metallic gaseous products of reduction at this stage through a free surface of the charge, while maintaining such free surface below the vaporization temperature of the volatile metal so as to prevent the escape thereof, and then at a subsequent stage subjecting the residue to an increased heat sufficient to fuse the same to slag and liberate said metal as vapor separate from the other gaseous products given off at the preceding stage.

Patent 859,134 relates to the use of the furnace shown in Fig. 4. It contains a smelting chamber *a*, and alongside of it a condenser *b* for liquid metal and the preheating chamber *c* to complete the roasting of the ore before it is fed into the smelting chamber through the opening *f*. The smelting furnace is an electric induction furnace, the charge, etc., surrounding the core *d*. In smelting lead-zinc ore, the furnace chamber contains molten lead in the two wells on either side, matte above the lead and slag above the matte, the slag ex-

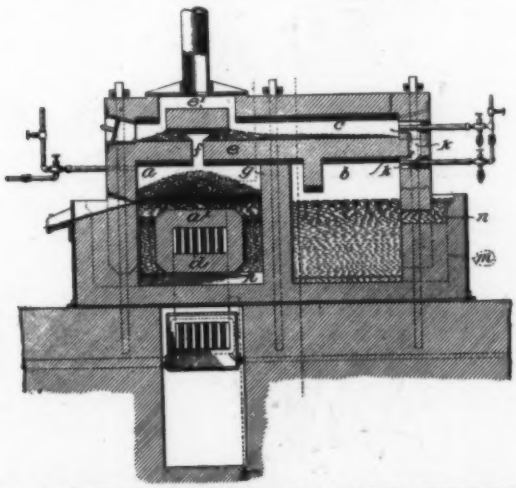


FIG. 4.—ELECTRIC INDUCTION FURNACE FOR ZINC PRODUCTION.

tending across the top of the bridge and electrically uniting the material in the two side wells. Above the slag the charge of ore is fed. When the charge has been introduced into the furnace the opening *f* is closed by means of the door e_1. The charge is now smelted in the absence of air, going through four stages of heating, the heat being applied at the bottom of the body of the charge and gradually spreading upwards, the portions nearest the slag being melted down first. During the first stage carbon monoxide is given off in considerable quantities with some zinc, but the upper portion of the charge is sufficiently cooled so that most of the zinc vapor liberated during this stage is condensed in or on the charge in the form

of dust which remains in the furnace chamber. After most of the CO gas been driven off, the second stage begins, in which the zinc vapor is given off freely, which is condensed in the chamber *b*. In the third stage the residues are fused to form slag and matte, and the fourth stage is that in which the fused charge is subjected to the maximum heat, the slag being superheated to expel substantially all the volatile metal. Patent 859,137 relates to essentially the same construction, but it also considers heating in a different way than by the induction principle, namely, by means of conductors dipping into wells which are in connection with the different molten baths in the compartments in a similar way as described above in the very first patent. (See also our Vol. IV., p. 319.)

Patent 859,135 (for the process) and 859,136 (for the apparatus) relate to the use of a furnace shown in vertical and horizontal cross-section in Figs. 5 and 6. It is operated by three-phase currents, three electrodes, 4, 5, 6, projecting vertically through the roof. In this case the purpose is not to produce directly liquid zinc but to produce first zinc dust, which is then resmelted into liquid zinc in another furnace. The electrodes are constructed of hollow carbon shells, which are filled with pulverized granular carbon feeding down through the hollow shells and resting on the slag bath underneath. The solid carbon shell does not touch the slag. The three flues 13 are provided for carrying off the gases due to reduction. Molten lead rests again on the bottom of the crucible. Above it there is a layer of slag and above that the charge is fed.

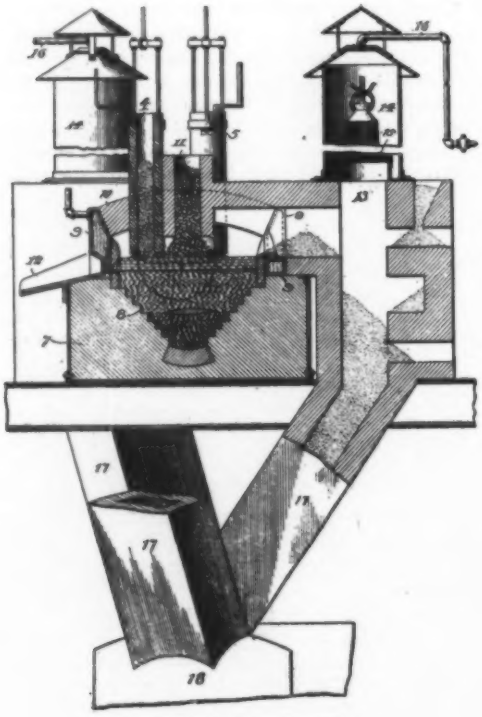


FIG. 5.—ZINC-DUST FURNACE.

The ores are first roasted down to about 6 per cent in sulphur and then mixed with fluxes, which will form a slag running about 40 per cent in silica, 30 per cent in lime and 15 per cent iron. Since the object is to get zinc dust, the charge is prepared so as to produce large quantities of other diluting gases, such as carbon monoxide. For this purpose the inventor adds to the ore a gaseous coal or limestone, and for the iron flux he uses iron carbonate (siderite), each of which will give off large amount of gases. He also wets the charge so as to get a considerable volume of steam. When the mixture is smelted in the furnace in the absence of air the metals are reduced and the excess of carbon serves to reduce the CO_2 to CO, to prevent the oxidation of the zinc dust produced. The lead sinks to the bottom of the crucible while the zinc volatilizes and passes off with the other dilute gases through the flues 13. The zinc vapors in these flues and in the stacks 14 are condensed in the form of zinc dust, which collects on the inside of the woolen bags 15 and generally throughout the flues, and then falls down to the bottom when the bags are shaken and passes through the flues 17 to the secondary furnace 18, where it is converted into spelter. The zinc dust produced by the first furnace is entirely protected from the air and retains as much of the original heat as possible when conveyed to the second furnace. The furnace charge is so regulated as to produce some metallic iron in the furnace. This may be done by having a large proportion of coal in addition to the amount of iron ore in the charge. The iron is reduced and floats on the molten lead, and its function is to decompose the zinc sulphide in the matte so that the matte is maintained as free as possible from zinc.

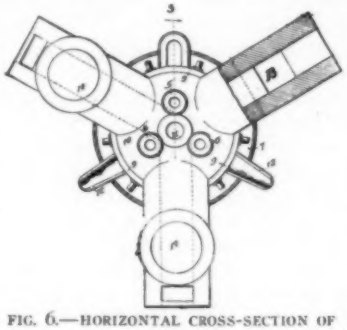


FIG. 6.—HORIZONTAL CROSS-SECTION OF FIG. 5.

Induction Furnace.—J. Härden, 861,031, July 23, 1907. Application filed Jan. 24, 1907. Assigned to Gröndal-Kjellin Co., Ltd.

When in the past attempts have been made to use polyphase currents by providing one annular bath for each phase, such attempts have been unsuccessful, according to the inventor, on account of the great loss by heat radiation, which is given as about 9 kw. per square foot of the bath surface. Moreover, as the annular fusion chambers have to be placed at a considerable distance from the primary coil, an arrangement of two or three fusion chambers greatly increases the magnetic leakage in the transformer, thus causing a low-power factor and inefficient operation of the generating plant. "According to the present invention, polyphase alternating currents are employed in the working of electric induction furnaces by directly combining therewith one or more transformers to convert the polyphase currents of the supply mains into single-phase currents in the induction furnace." Diagrams are first given of feeding an induction furnace with two-phase currents, and then, by means of the well-known transformation of C. F. Scott from three-phase to two-phase supply the connections of an induction furnace to a three-phase supply are also shown. (To convert polyphase currents into single-phase simply by means of transformers, that is, without the aid of rotary machinery, is utterly impossible; see, for instance, C. P. Steinmetz, *Alternating Current Phenomena*, 3d edition, p. 460. It must therefore be concluded that with the arrangements shown in the patent the polyphase supply system must become unbalanced.)

Charging Electric Furnaces.—W. H. Huffman, 860,477, July 16, 1907. Application filed Feb. 15, 1907.

The object is to so charge an electric furnace of the Acheson type (graphite furnace) that a clear plane of demarcation is obtained between the furnace charge proper and the surrounding or embedding material. In charging the furnace some embedding material is first placed at the bottom and then longitudinal walls are built extending throughout the length of the furnace, so as to form a receptacle or channel for the charge. The charge is then filled in and the embedding material is filled in on the outside and the dividing walls are then withdrawn.

Electric Furnace.—Paul L. T. Héroult, 858,718, July 2, 1907. Application filed April 13, 1906. Assigned to the Soc. Electrometallurgique Française.

Details of construction of an electric furnace with one carbon electrode, the other terminal being formed by the carbon crucible itself. The construction is shown in vertical and horizontal cross-section in Fig. 7. The body A of the furnace is of carbon and bound by a surrounding jacket or rings of iron, steel or the like. The current passes from the electrode B through the charge and then through the cast iron piece C to the cable D. Ordinarily the jacket is made of iron or steel, forming a continuous ring of magnetic material so that a magnetic flux is set up within this ring. In order to eliminate it the iron portion E of the jacket is not continuous around the periphery, but is broken at one or more points, the edges being connected by a strip F of copper or other non-magnetic material. This reduces very largely the impedance of the furnace when operated with alternating current. A more perfect contact than usual between the body of the furnace and the cast-iron base is secured by providing long forked pins H, which extend from the base upward into the carbon block and the lower ends of which are cast into the base plate C so as to make perfect contact therewith. For making a good connection between the cable D and the plate C a special contact block J of copper is provided, about which the plate C is so cast as to effect an intimate welded union.

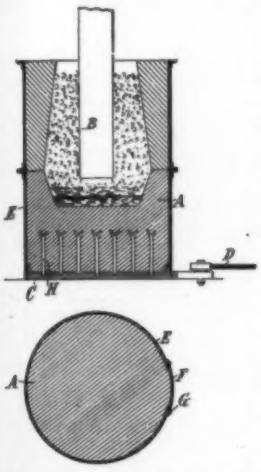


FIG. 7.—VERTICAL AND HORIZONTAL SECTION OF ELECTRIC FURNACE.

Electrodes for Electric Furnaces.—F. von Kügelen and G. O. Seward, 858,400, July 2, 1907. Application filed Jan. 31, 1905.

When carbon electrodes are used in electric furnaces it has been found impossible to prevent a portion of carbon from combining with the charge. Even if the electrodes plunge into the slag above the metal, carbon comes into the latter. The slag always contains oxides of the metal to be produced or refined, and these oxides are reduced by the carbon from the electrodes to a carbide which spoils the metal below. The present inventors use metallic electrodes which are water cooled. They describe the application of the process to the refining of ferro-chrome containing originally a high percentage of carbon—say 6 to 8 per cent. A crucible is made of carbon-free materials, such, for example, as chrome ore. The charge is made up of a refining slag of chrome ore and lime, under which the metal to be refined is melted. The charge is first melted by temporary carbon electrodes. These are used for so short a time as to introduce only a negligible quantity of carbon. Then water-cooled steel pencils or electrodes are introduced into the molten charge. The temperature of that portion of the charge in contact with the pencil is reduced to a point so low that it will not materially attack or melt the pencil. The current passes from one pencil to the slag and thence to the other pencil and the metal at the bottom of the crucible is gradually refined by the action of the oxide slag on the carbon of the metal.

ELECTROLYTIC PROCESSES.

Electrolytic Production of Sodium Oxide.—C. F. Carrier, Jr., 859,431, July 9, 1907. Application filed June 9, 1906. Assigned to Elmira Electrochemical Co.

As now prepared, the oxides of sodium are made by the oxidation of previously produced metallic sodium. The object of the present process is to make them in one operation from sodium chloride. The cell for this purpose is shown in Fig. 8, the anodic compartment 7 being at the left and the cathodic compartment 15 at the right, both being in connection with each other through the layer of fused lead 30 at the bottom. The electrolyte 9 in the anode compartment is sodium chloride, the anodes 12 consisting of Acheson graphite. The electrolyte 17 in the cathodic compartment is fused sodium hydroxide with an oxidizing agent like sodium nitrate. By means of the burners 29 the furnace is heated to over 350° C., and lead is introduced into the pan to form a liquid seal at the bottom of the electrolytic compartment. After the electrolytes have been filled in and the circuit has been closed, the electrolytic decomposition of the sodium chloride in the compartment 7 results in chlorine being set free at the anode and carried off through the port 14, while simultaneously the sodium set free at the fused-lead cathode alloys with the lead. By means of the propeller

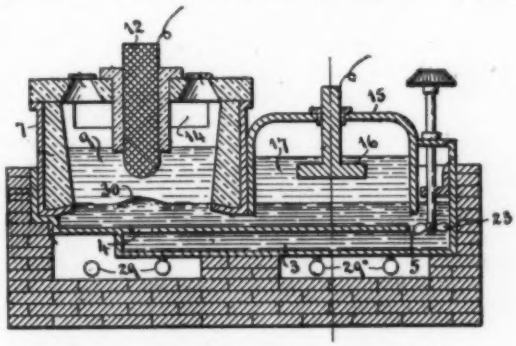


FIG. 8.—ELECTROLYTIC PRODUCTION OF SODIUM OXIDE.

23 a rapid circulation of the lead is produced through the two openings 4 and 5, by means of which the upper electrolytic compartments are in connection with the lower longitudinal compartment 3. In this way the sodium-lead alloy is carried away from the compartment 7 to the compartment 15, which is in form of a bell, completely closed except at the bottom. Here it acts as anode and sodium passes into the electrolyte NaOH. At the cathode 16 of nickel or iron, sodium would be set free if the electrolyte consisted simply of NaOH, but since it contains an oxidizing agent, sodium oxides are produced according to the following equations:

$$NaNO_3 + 3NaOH + 8Na = 6Na_2O + NH_3 \text{ (I.)}$$
$$NaNO_3 + 3NaOH + 2Na = 3Na_2O_2 + NH_3 \text{ (II.)}$$
$$Na_2O + \text{air} = Na_2O_2 + \text{nitrogen.}$$

Reactions I. and II. take place simultaneously, their relative preponderance depending upon the conditions of the operation. The product, as withdrawn from the cathode compartment, will contain both Na_2O and Na_2O_2. The Na_2O may be converted into Na_2O_2 by heating in an oxidizing atmosphere, air free from moisture and carbon dioxide being most suitable. Sodium nitrate is added from time to time.

Manufacture of Steel Dies.—G. F. Diez, 857,926, June 25, 1907. Application filed Feb. 5, 1907.

The purpose is to produce steel dies directly from the object to be reproduced without the necessity of engraving the die. The method, therefore, consists essentially in depositing iron electrolytically directly upon the surface to be reproduced and then in so treating the iron matrix thus formed as to convert it into a grade of steel best suited for producing positive impressions in the material chosen for the copies of the original. The process is especially applicable to the reproduction of the finest type of engraving where faithfulness to the copy is important. The inventor produces upon the body to be copied a film of exceeding fineness, so fine and homogeneous in texture as to produce a deposit which from beginning to end is white like tin and at the same time so free from corrugations and

roughness or crystalline structure as to have the softness of silk to the touch. This is later converted into steel of any degree of hardness. The plating bath is made of 1,000 cubic centimeters of a saturated aqueous solution of iron sulphate and 125 cubic centimeters of a saturated aqueous solution of sodium bicarbonate (the sodium bicarbonate being added to the iron sulphate in two halves, the second half being added after 12 hours, and the solution being ready again 24 hours later). The anode is made of iron. A voltage of 0.7 to 1.5 is used, the density of the bath is 18° to 20° Baume and the temperature 21° C. The object to be coated is hung by a fine copper wire at a distance of 10 to 25 cm. from the anode. Under normal conditions a matrix thick enough for type or for seals to stamp on leather or paper can be obtained in from three to four days, depending on the dimensions, while a matrix to be attached to a block of steel and used as a die to stamp on metals will require from ten to fifteen days. In order to separate the deposited metal from the original or master, it is taken from the bath and washed with distilled water, after which, if the mold be metallic, the mold and matrix may be heated and may then be separated by inserting a thin blade between them and carefully prying them apart. If the original be of rubber, a slight heating will be sufficient, while when waxy compositions are used an immersion in boiling water is all that is necessary. In order to convert the soft iron matrix into steel of the required hardness the inventor takes charcoal powder 8 parts, steel filings 1¾ parts, potassium ferrocyanide ¼ part. To this mixture he adds a sufficient quantity of oil to form a paste. The several ingredients are all ground together in a mortar, and the oil is added in small quantities from time until the desired paste is formed. The crucible is half filled with the paste and the matrix is first wet with oil and then covered with powdered charcoal until a coating is formed thereon, and is then placed in the crucible with the matrix face down, after which the crucible is filled with the paste and pressed down and covered with plaster. The crucible is then placed in an oven and

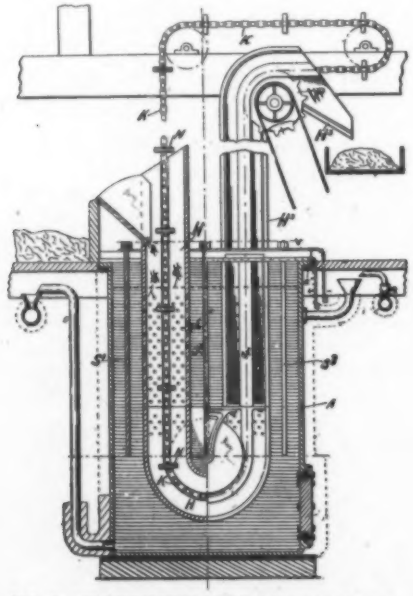


FIG. 9.—ELECTROLYTIC DETINNING APPARATUS.

maintained at a red heat from 8 to 10 hours, when the plate will be found to be converted into the finest steel, after which it may be tempered and is ready for use for stamping metals, and will stand the pressure of the most powerful presses.

Electrolytic Detinning.—M. Leitch, 859,565 (apparatus patent) and 859,566 (method patent), July 9, 1907. Application filed Feb. 17, 1906. Assigned to American Can Co.

The apparatus is about the same as described in the former patent of the inventor abstracted and illustrated on page 103 of our March issue. The object is to provide a continuous method of detinning in which all manual labor is eliminated except that required to feed the tin plate scrap into a hopper and collect the tin powder from the cathode plates. As shown in Fig. 9 the electrolytic tank A contains a U-shaped basket or tube H, the sides of which are perforated. Within this U-shape basket the tin scrap to be detinned is placed, and is moved down and up by means of the T-bars NN fastened to the conveyor chains K. As is indicated in the illustration the scrap is finally discharged through the chute H_3 into the washing tanks through which it is also carried automatically. The electrolyte is a solution of caustic soda heated to about 75° C., and three cathodes $S_1S_2S_3$ of iron are provided on which the tin is deposited in spongy form.

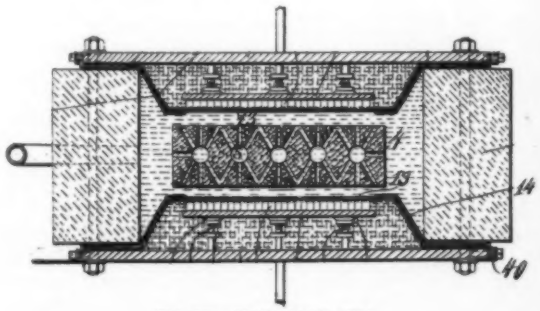


FIG. 10.—DIAPHRAGM CELL.

Detinning.—R. S. Wile, 859,792, July 9, 1907. Application filed Oct. 19, 1906.

The process consists of two steps, carried out in different vessels. The first step is the solution of the tin from the tin scrap by a purely chemical method, the second step is the electrolytic deposition of tin from the solution. The tin is removed from scrap of tine plate by a solution of 20 per cent of stannic chloride containing from 0.5 to 5 per cent of hydrochloric acid. By the absorption of the tin in the stannic chloride the latter is changed into stannous chloride. This process is carried out in an apparatus such as used for pickling sheet iron or steel. The stannous chloride solution is then run off into the electrolytic vat and electrolyzed with carbon anodes with a cathodic current density of 1 amp. per square inch. At the anode the stannous chloride is oxidized back to stannic chloride, while at the cathode tin is deposited in form of silvery white crystals. (In its principle the process is quite analogous to the old Hoepfner copper process.)

Diaphragm Cell for Sodium Chloride Electrolysis.—G. A. Gabriel, 12,672, July 16, 1907. Application filed May 29, 1906. Assigned to Bleach & Caustic Process Co.

This is a modification of the Hargreaves cell for the electrolytic production of chlorine and caustic soda. The general construction of the cell is shown in Fig. 10, representing a horizontal section. The Acheson graphite anodes 4 are provided with vertical and horizontal passages to permit thorough circulation of the electrolyte; 11 is a foraminous screen upon one side of which is placed the porous diaphragm 13. This screen 11 forms an active portion of the cathode. To restrict the active surface of the screen and diaphragm, the insulating material 14 is provided, consisting of heavy oiled silk, mica or hard vulcanized rubber. The outside of the screen 11 is provided with draining means to carry off the caustic soda formed, as was described and illustrated in our Vol. IV., page 322. Means for carrying off the products and controlling the level of the electrolyte in the anode compartment are also described. There are no less than 30 claims.

Cathode for Mercuric-Salt Solutions.—H. S. Hatfield, 860,657, July 23, 1907. Application filed Aug. 20, 1906.

As cathodes for the electrolysis of solutions containing mercuric salts, substances of the iridium group are recommended. The members of this group are iridium, tantalum, niobium, vanadium, silicon and magnetite. The inventor prefers iridium in the form of rolled sheet and hardened by sand blast.

Electrolytic Refining.—Anson G. Betts, 857,378, June 18, 1907. Application filed Feb. 20, 1905.

In electrolytically refining metals, such as copper, to which the anode residue does not ordinarily stick, the anode area in a tank finally becomes reduced while considerable anode copper still remains undissolved, thus increasing the anode current density, and in certain parts of the cathode surface the cathode current density. As the refinery contains a large number of tanks in the same electrical series it is not desirable to reduce the current passing for the sake of maintaining a proper current density in one or a few tanks to make pure copper, and as a consequence the anodes are removed from the tanks as scrap before they are well used up. Instead of this arrangement the inventor uses one or more shunts, with a switch for each, to shunt parts of the current around any undivided tank, or set of a few tanks, when the anode area becomes reduced. When the tank is first used the shunts are disconnected and the current entirely passes through the electrolyte. Toward the last, when the anode area begins to diminish considerably, a shunt is connected in circuit, and when only a small part of the anode surface is left all the shunts are used, when the electromotive force at the bus-bars is particularly the same as it was at first, and the small anode area remaining is undergoing electrolysis at the same rate as on the start. In refining metals, such as lead, containing 5 per cent of impurities, this process has the advantage that no copper, etc., will dissolve in the electrolyte which would happen if the current density and therefore the e. m. f. would become too high.

Nickel by Electrolysis.—H. H. Dow, W. S. Gates, A. E. Schaefer, 857,927, June 25, 1907. Application filed April 18, 1906. Assigned to Ontario Nickel Co., Ltd.

See the abstract of the patent relating to the recovery of nickel contained in basic nickel precipitates in the department "Recent Metallurgical Patents."

BATTERIES.

Battery Electrode.—T. A. Edison, 857,929, June 25, 1907. Application filed March 30, 1905. Assigned to Edison Storage Battery Co.

In the Edison iron-nickel storage battery the active mass of nickel hydroxide of the positive electrode is admixed with a flake-like conducting substance to reduce the resistance. Formerly flake graphite was used for this purpose. In course of a long time, however, flake graphite undergoes a change in its contact resistance and the capacity of the battery is diminished. Edison now uses flakes of a nickel-cobalt alloy containing, say, 60 per cent of cobalt and 40 per cent of nickel.

Battery.—F. A. Decker, 857,606, June 25, 1907. Application filed March 30, 1904. Assigned to Decker Electric Manufacturing Co.

The chief feature of this battery is the use of a fragmental zinc electrode, consisting of zinc balls, granular scrap or the like within a porous cup.

Battery.—F. A. Decker, 857,607, June 25, 1907. Application filed Jan. 16, 1906. Assigned to Decker Electric Manufacturing Co.

Means for conveniently and compactly connecting a number of cells together for facilitating the operations of filling and emptying them. Claim 1 specifies "a plurality of crates arranged in parallel with corresponding ends connected together, each crate having bottom strips of less width than the ends and side strips disposed at different elevations permitting juxtaposed side strips of joined crates to lie one above the other, in combination with a series of cells in each of said crates, and conduits secured to the bottoms of the envelopes of the cells and disposed on either side of the bottom strips thereof, the conduits communicating with the interiors of the respective cells."

Battery Containing-Cell.—G. H. Stout, 842,123, Jan. 22, 1907. Application filed Aug. 13, 1903.

Claim 1 specifies "a cell comprising a casing or cell proper having a series of slots or grooves on opposite sides, a series of elements received in said slots to form one electrode, and a plate arranged in the bottom of the cell to form the other electrode."

RECENT METALLURGICAL PATENTS

COPPER.

Bessemerizing Copper Matte.—Under ordinary conditions, when the slag formed in the converter has combined with all of the iron present in the matte and has collected it with other metals and the silica, the slag is removed from the converter, and with whatever proportion of iron oxide or other oxidized metal it may contain, is returned to the reverberatory or blast furnace to be used as a flux. Richard L. Lloyd, of Cananea, Mexico (857,816 and 857,817, April 30, 1907), proposes to maintain the slag in the converter or return it thereto until such time as its acidity, or at least its power of collecting bases, is exhausted; in other words, until it is practically saturated with the oxides. When in this condition it is much more efficacious as a flux, and a considerable economy (both in the converter and in the furnace) is secured in the use of the silicious ore, the expense of obtaining which for the reduction process is an important item in the cost of the refined copper. While in ordinary practice the moment of withdrawal of the slag is based on the condition of the copper matte, it is based in Mr. Lloyd's process on the ability of the slag itself to collect bases.

The process of Lloyd may best be seen from Fig. 1. A and A^1 are two feeding barrels, A being charged with copper matte and A^1 with the slag or flux. The mattes and the slag

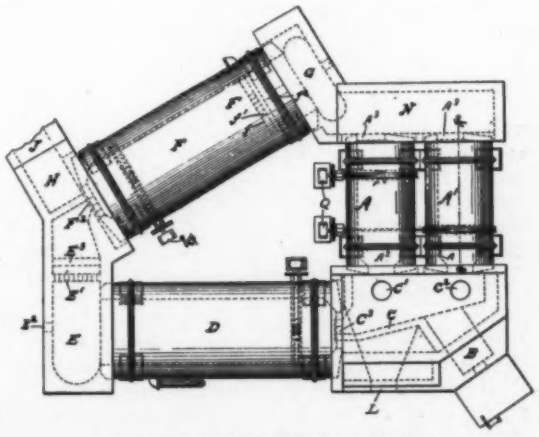


FIG. 1.—BESSEMERIZING COPPER MATTE.

are fed to the hearth C, and at B a furnace is shown which may be used in the initial operation of the plant to heat the materials and start the process. From the hearth C the matte and slag pass through the spout C^3 into the primary converter D, where the matte is fused and blown. The iron and other impurities go into the slag, while the copper sulphide settles to the bottom of the converter. From the converter D the slag and copper sulphide are discharged into the hearth E. A siphon dam, consisting of the wall E' with perforations at its

bottom, and an abutment E^3 higher than the perforations and beyond them, forms the outlet from the hearth E; while an opening E^2, a little higher than the abutment E^3 forms a slag outlet, the operation of the arrangement being such as to completely separate the copper sulphide from the slag. If the acidity of the slag be exhausted it may be at once returned to the furnace for use as a flux or otherwise disposed of, while if it is still acid it may be returned to the converter D through one of the openings C^1, C^2, in the hearth C. The copper sulphide passes by a suitable spout F^2 into the second converter F, where it is again subjected to blast and the remaining sulphur blown out of it. At the exit end of the second converter is a siphon dam formed of the wall *f*, having the perforations f^1 and the wall f^2. The copper being heavier than the copper sulphide sinks and passes through the perforations f^1 and over the wall f^2, while the copper sulphide is kept in the converter until the sulphur has been completely oxidized and driven off. From the converter F the copper passes into the hearth G, where it may be poled or otherwise further refined. The hearth E is impervious to the blast, which proceeds from the converter D through the feeding barrels A, A^1, the flue N and the hearth G (which is the coolest point in the operation) into the converter F, the flue H to the flue J, and thence to the stack. In this process the flow of copper sulphide from the first to the second converter, as well as the flow of metallic copper from the second converter into the hearth is practically continuous.

Pyritic Smelting.

Use of Electrically Heated Air.—In the pyritic smelting of ores which are deficient in sulphur so that the heat of oxidation is insufficient to maintain or complete the reaction. Henry Arden (860,512, July 16, 1907) endeavors to supply the deficiency with the aid of hot air heated in an electric furnace. This is a resistance furnace, the resistors being carbon bars coated with a refractory layer of carborundum, applied as a paint with sodium silicate as a binder. Another excellent resistor, particularly where currents of comparatively high voltage are employed, is said to be bars of carborundum consisting of grains bound by sodium silicate. Such resistors will not be oxidized by the air. It is also advantageous to introduce with the hot air a heavy hydrocarbon or the vapor of a hydrocarbon, the oxidation of which also supplies heat. The inventor vaporizes the hydrocarbon in an independent furnace, preferably in an electric resistance furnace, and conveys the vapors through the furnace employed for heating the air and thence into contact with the ore. The quantity of hydrocarbon vapors may be easily regulated by suitable automatic feed, to meet the two purposes of its introduction, viz.: to diminish the amount of oxygen under a given pressure of the blast or to increase the heat thereof.

Smelting Furnace.—Although not a system of pyritic smelting, the method of W. Kemp (846,216, March 5, 1907) may be mentioned here. The chief feature of the furnace is that the ore to be smelted is not mixed with the fuel. The ore chamber is longitudinal and situated in the center of the furnace. The fuel is introduced separately into two longitudinal fire places running at the bottom of the furnace on both sides of the ore chamber. The fuel is supplied with an up-blast from the bottom and a down-blast from the top. It is fed cold into the fuel chambers and burns there in immediate contact with the ore. A blast is also introduced into the ore chamber above the fuel chambers. In the case of a sulphide ore much of the smelting heat is obtained from burning the sulphur. The chief saving of fuel is, however, due to the fact that no carbon monoxide can escape, as is the case if the carbon is mixed together with the ore. In the present case the carbon is completely burned to carbon dioxide.

Iron.

Blast Furnace Top.—C. H. Clark (857,158, June 18, 1907) proposes to remove as much as possible of the charging mechanism from the top of the furnace and to operate the mechanism with the aid of an endless cable and with suitable reducing gear directly from the hoisting engine, so that, for instance, the furnace top will turn one-quarter or any other desired fraction of a complete rotation during the lifting of a car of stock.

Charging Apparatus.—G. R. Johnson (853,016, May 7, 1907) wishes to do away with the old bell and hopper arrangement altogether and substitute a charging apparatus for which he specifies in claim 1 "the combination of a blast furnace, a revoluble spout extending into said blast furnace, a valve hinged to the lower end of the spout, chains connected with the valve, pipes carried by the spout through which said chains extend, means for operating the chains in a direction to close said valve to cause it to support a mass in the spout, said valve when opened serving to release said mass, and a second valve supported independently of the spout and closable against the upper end thereof above the mass therein and to constitute a seal for said spout."

Tuyeres.—J. W. MacDonald (860,983, July 23, 1907) refers to the disadvantages which result if in blast furnaces having a series of water-cooled metal jackets in their walls the tuyeres form part of the water jackets. He proposes to arrange the tuyeres in specially constructed water-cooled chambers independent of the metal jackets and designed to be separately removable from the furnace walls.

Agglomerating Blast Furnace Dust.—W. Schumacher (859,411, July 9, 1907) patents a binder for agglomerating blast furnace dust or other comminuted or pulverulent substances. Very finely ground silicious material is mixed with lime and with the substance to be agglomerated and then exposed to the action of steam under pressure. The best results are obtained if the silicious material and the lime are ground together so that the silicious material in being broken up will exert a grinding action on the lime and the small particles of silicious material will be covered with lime dust.

Flux.—J. Davies (858,582, July 2, 1907) recommends the following flux or "physic" for the treatment of pig iron: Charcoal 2/16 per cent, rock salt ¼ per cent, limestone 1/3 per cent, plumbago 3/32 per cent, soda 2/3 per cent, manganese ¼ per cent, nickel 6¼ per cent. The aggregate of these ingredients is nearly 8 per cent, the remaining 92 per cent being molten iron to which the flux is added. The effect is said to be that the impurities in the iron are eliminated, that the metal is free from slits and blow-holes and that it has a greater tensile strength and density.

Gold and Silver.

Pressure Filter.—In a pressure filter patented by A. E. Davis, of Johannesburg, Transvaal (859,749, July 9, 1907), a series of vertically disposed filter plates are arranged inside a receptacle into which the slimes or other material to be filtered are passed. The separation of the solid matter is effected by the deposition of the particles of solid matter on the superfices of the filtering cloth which covers the surfaces of said plates, the separated liquid flowing along grooves or flutes in the surfaces of the plates to a common outlet or delivery. The object of the construction of the filter plates is to facilitate the discharge of the separated solid matter. The filter plates are provided on both sides with parallel and vertical grooves, and are made to taper from the top to the bottom and are enclosed in the filtering cloth. With the object of preventing the settlement or deposition of solid matter on any portion of the filter plate adjacent to the sides or top of the filter, the sides or top of the filter plates are made with a non-filtering strip or ungrooved portion. When the liquid has been pressed into the grooves and drawn off into a delivery pipe, the solid matter remains outside. When the filter box is practically full with solid matter its bottom door is opened and water is pressed into the filter plates in the reverse direction to that of the previous filtering flow. The

solid cake is thereby pressed away from the cloth and the cake drops out of the box.

Filter.—Quite a similar construction is patented by W. T. Janney and G. H. Dern, of Mercur, Utah (857,467, June 18, 1907). Instead of filtering plates they employ, however, filtering sticks. They thus have a large number of small units, each of which can be readily repaired without interfering with any other unit. The filtering sticks are made of wood of conical or tapering form and provided with an annular series of longitudinal grooves. The stick is covered by a canvas bag and the liquid is separated from the solid either by suction from the inside of the filtering sticks or by pressure on the outside. The liquid is drawn off through the grooves in the filtering sticks. The dislodging of the cake from the canvas bags is easy on account of the tapering form of the sticks.

Chlorination Barrel.—W. J. Armbruster patents a chlorination barrel, the feature of which is the construction of the tube through which the chlorine gas is introduced into the barrel. Instead of using a stationary delivery pipe or goose-neck, he employs a revolving pipe having an intake-end fixed within an opening disposed about the axis of rotation of the barrel. The joint interposed between the stationary feed pipe and the hollow trunnion of the barrel (through which trunnion the fixed intake-end passes) is in the nature of a removable sleeve feathered to the feed pipe, and has one end in forcible, yet movable engagement with the adjacent end of the trunnion. One of the trunnions of the barrel is hollow, the passage therein receiving the fixed or intake-end of the chlorine delivery pipe p. The latter extends in the form of two branches, p' p'', disposed along the inner surface of the barrel, the branches being secured to the adjacent head of the barrel by means of straps 4. This disposition of the branches p' p'' prevents the pulp at any time from backing into the intake-end p. The outer end of the hollow trunnion in the present construction terminates in an enlarged circular head 5, whose outer face is provided with an annular groove for the reception of a correspondingly shaped ring 7 formed on the adjacent face of one of the terminal flanges 8 of a rotatable sleeve 9, feathered to move longitudinally along the adjacent end of the stationary supply or feed pipe 10.

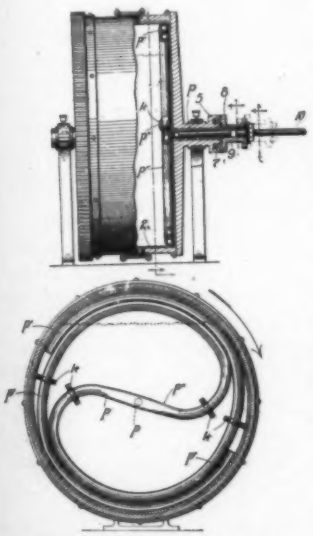


FIG. 2.—CHLORINATION BARREL.

NICKEL.

Treatment of Ores Containing Nickel, Copper, Cobalt and Iron.—A patent of R. W. E. MacIvor (859,776, July 9, 1907, assigned to Metals Extraction Corporation, Ltd.) relates to the treatment of ores containing nickel, whereby the latter metal is first concentrated into a matte or regulus, which is afterwards subjected to a process of slow roasting to convert the metallic sulphides composing the matte into the corresponding sulphates. The method of forming the matte or regulus from the ore varies with the character and composition of the latter, but the agent employed is in every case sodium sulphide containing an excess of sulphur over the proportion contained in the normal sulphide Na_2S. In the case of sulphide arsenical ores a previous "dead" roast is essential in order to get rid of the arsenic, whereas with ores of the double silicate of nickel and magnesia type (garnerite) no preliminary roasting is necessary before mixing with the sodium sulphide. To carry out the operation of forming the matte the roasted or unroasted ore, as the case may be, is thoroughly mixed with from one-fourth to one-half its weight of the sulphide of sodium and not more than 5 per cent of carbon, preferably ground anthracite coal. This mixture is made into blocks or bricks with water, and after being dried sufficiently to allow of handling, they are smelted in a reverberatory furnace. The temperature is gradually raised to and maintained at the point of complete decomposition of the sulphate of iron present in the roasted mass, the sulphate of nickel and copper being left unaltered. These latter salts are obtained in solution by throwing the hot mass from the furnace into the water acidulated with sulphuric acid to an extent dependent upon the copper in the ore under treatment. This solution is allowed to settle, drawn off from the oxide of iron and other sedimentary matter and treated by known methods for the recovery of its copper contents. After the extraction of the copper the nickel and cobalt are precipitated from the solution by milk of magnesia or lime, or by soda, in the form of oxides, and these are collected and dealt with by known methods.

Nickel from Basic Nickel Precipitates.—When basic nickel precipitates, such as the hydrate and the carbonate, are treated with free halogens, such as chlorine and bromine, only a part of the nickel is dissolved to soluble haloids, and the balance remains insoluble, having been oxidized by the halogen to black nickelic hydrate. The latter remains insoluble unless it be dissolved in acid, or unless a very strong halogen be used. In the latter case some of the nickelic hydrate (NiO_2H_2) may be dissolved, giving a dark-colored unstable solution, which readily decomposes upon standing or heating, thus giving off the halogen and reprecipitating the black nickel hydrate. The action of the halogen upon the nickel hydrate is as follows:

$$3NiO_2H_2 + Cl_2 = NiCl_2 + 2NiO_2H_2,$$

so that only one-third of the nickel is rendered soluble. In order to dissolve all of the nickel by means of a halogen some reducing agent must be present to prevent formation of the nickelic hydrate. H. H. Dow, W. S. Gates and A. E. Schaefer (857,927, June 25, 1907, assigned to Ontario Nickel Co., Ltd.) have found that either sulphur in the elementary state, preferably as ground flowers of sulphur, or in form of some compound like sulphur chloride serves the purpose very well. The reaction when the basic nickel precipitate is treated with free halogen in the presence of sulphur is as follows:

$$4NiO_2H_2 + S + 3Cl_2 = 3NiCl_2 + NiSO_4 + 4H_2O.$$

Compounds of sulphur act in a similar way, but in whatever form sulphur may be employed the nickel obtained by this method is a mixture of chlorides and sulphates, and if it is intended to win the nickel afterwards by electrolysis with carbon anodes it is necessary to remove the nickel sulphates, since the carbon anodes would be destroyed. This is possible in one of the following two ways: either by the addition of CaO_2H_2, when the halogen is acting on the nickel precipitate in the presence of sulphur, or by the addition of the proper amount of calcium chloride or calcium bromide with a subsequent precipitation of calcium sulphate. In the electrolysis the halogen, for instance, bromine, is set free at the anode, while simultaneously nickel is deposited on the cathode. The bromine is then returned to the dissolving apparatus, where it acts upon more basic nickel precipitate in the presence of sulphur, etc.

TIN.

Removal of Iron.—H. Brandenburg (859,594, July 9, 1907) patents a process for separating the iron and tin contained in the intermediate products of the treatment of tin ore, that is to say, in waste products, residues, tin-containing slag, ashes,

scrap, soot and the like. The waste products, etc., are smelted with silicate of tin in presence of a basic flux. The silicate of tin is either specially prepared for the purpose or is obtained from the slags produced during the extraction of tin by former processes. The result is that the tin—both of the waste products and of the silicate—appears in metallic state, while the iron combines with the silicate. Any oxide or dioxide of tin that may be dissolved in the slag reacts in a similar manner.

Smelting Process.—H. Brandenburg (859,184, July 2, 1907) remarks that the reduction of tin ore in a smelting furnace with the formation of slag has the disadvantage that dioxide and oxide of tin dissolve in the slag, from which they can be extracted only with difficulty and expense. He recommends to prevent the oxide and dioxide of tin from passing into the slag by effecting the reduction at temperatures which preclude the possibility of the formation of any slag. The reducing process in such case takes place in such a manner that metallic raw or unrefined tin is obtained embedded in the loose, and not slagged, silica of the ore.

Detinning.—See abstracts in Analysis of Current Electrochemical Patents in this issue.

Zinc.

Charging Retorts.—L. W. Kirk (857,074, June 18, 1907) patents a machine for charging retorts which consists essentially of an injector tube *a* of metal through which the comminued charging material is forced into the retorts, a hopper *d* connected with the tube for supplying the charge, and a nozzle *c* connected with the tube and delivering a jet of steam or air behind the material discharged from the hopper into the tube. The material is thereby forced in a compact stream into the body of the retort, and at the same time the material is drawn from the hopper into the tube by reason of the injector-like action of the jet.

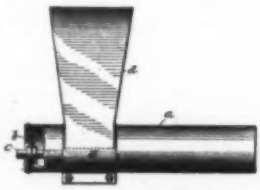


FIG. 3.—MACHINE FOR CHARGING RETORTS.

Electric Zinc Furnace.—See a number of recent patents granted to Fred. T. Snyder and abstracted in the Analysis of Current Electrochemical Patents in our present issue.

Roasting.

Chloridizing Roasting.—Refractory ores are subjected to a chloridizing roast according to R. McKnight (858,667, July 2, 1907) as follows: An intimate mixture is made of the ore to be roasted with salt, the ore being reduced to about 30-mesh. Practically salt to about 10 per cent of ore is added. The mixture is roasted with agitation in a revolving cylinder, and there subjected to a large and continuous supply of superheated oxygen-containing gas, such as steam or air, having a temperature of about 500° C. without any outside source of heat. The superheated air or steam is introduced as a large jet or blast. With a refractory ore containing zinc sulphide, lead sulphide, iron pyrites and silica, the following reactions are said to take place:

(1) $ZnS + 2O_2 = ZnSO_4$.
(2) $ZnSO_4 + 2NaCl = Na_2SO_4 + ZnCl_2$.
(3) $PbS + 2O_2 = PbSO_4$.
(4) $PbSO_4 + 2NaCl = PbCl_2 + Na_2SO_4$.
(5) $4FeS_2 + 15O_2 + 2H_2O + 16NaCl = 4FeCl_3 + 4HCl + 8Na_2SO_4$.
(6) $4FeCl_3 + 3O_2 = 2Fe_2O_3 + 6Cl_2$.
(7) $4NaCl + 2SiO_2 + O_2 = 2NaSiO_2 + 2Cl_2$.
(8) $2Au + 3Cl_2 = 2AuCl_6$.

Reactions 1, 3, 5, 7 are primary; 2, 4, 6, 8 are secondary.

When the ore is removed from the roasting furnace it is passed into a suitable amalgamator while it is still hot.

Magnetic Roasting.—In what is known as a magnetic roast the ore is brought into such a condition that it may be afterwards easily separated by magnetic concentration into magnetic and non-magnetic material. A difficulty which has been experienced in such processes has been due to too high a temperature, whereby the ore particles have been brought into slightly fused condition so as to cling together. The magnetic material is thereby contaminated and the object of the roast is neutralized. A. R. Wilfley (859,420, July 9, 1907) roasts such ore by causing it to travel downward by a current of hot air instead of in opposite direction to such a current as is the usual custom. The roasting furnace is shown in Fig. 4. Hot air produced on the fire-box of the right-hand compartment passes upward, around the top of the dividing wall downward, together with the finely divided ore introduced through the top of the furnace. The course of the hot air is indicated by the arrows. When the roasted ores come down they drop on the water-cooled portion 15, 16 being composed of fire-clay and 17 and 19 being also water-cooled portions. The particles of the ore are thereby cooled so that the magnetic and non-magnetic particles will not stick together; they fall into vibrating trough 21, which is also water jacketed. Provision is made for imparting a longitudinal reciprocating movement to the trough whereby the roasted ore is caused to travel out of the furnace.

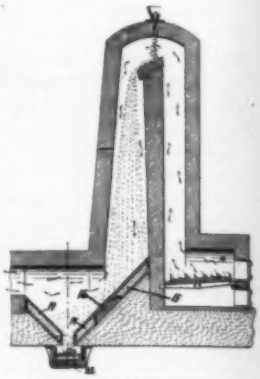


FIG. 4.—ROASTING FURNACE.

Aluminothermic Process.

Production of Manganese.—In the well-known Goldschmidt process as carried out heretofore the manganese compound Mn_3O_4, representing the lowest state of oxidation, is reduced by means of aluminium to metallic manganese. However, on account of the slow process, the output is comparatively small. It has now been found by Dr. Hans Goldschmidt and Dr. O. Weil (860,799, July 23, 1907) that a far better output up to 90 per cent and more may be obtained by adding to the Mn_3O_4 a relatively small quantity of a higher oxide, such as Mn_2O_3 of MnO_2. In general, very satisfactory separation of the manganese takes place if there is used a mixture of manganese compounds containing about 8 per cent of active oxygen (that is only 1 per cent more than in Mn_3O_4). A suitable mixture for the manufacture of manganese is the following: 73 kilograms of Mn_3O_4 and 5 kilograms MnO_2, both in the form of a powder, and 22 kilograms finely granulated or powdered aluminium are mixed and then ignited. When the process of reduction takes place new masses of the mixture are added, as already known in the Goldschmidt process.

Production of Boron.—K. A. Kuehne (861,129, July 23, 1907) states that those metals or alloys which cannot be produced or isolated separately in reguline form by the simple aluminothermic process can be reduced by aluminium and chlorate of potassium without the application of external heat. For instance, for the production of boron a mixture is prepared of two parts of finely divided aluminium, 5 parts of chlorate of potassium and 3 parts of a comopund of boron. The mixture is placed in a crucible and ignited by touching it with a red-hot iron rod or a strip of magnesia. The mixture begins to burn and melts together at a white heat into a thin liquid boiling mass, which consists, after cooling, of two easily separated layers. The upper layer consists of melted clay, sometimes set or stiffened with crystals, while the lower layer consists of a regulus of aluminium, which contains crystallized boron. The boron is then isolated by dissolving the aluminium in dilute acid.

Lead-Lined Hose

A flexible metallic hose which should find considerable application in chemical and metallurgical works and in allied industries, is being manufactured under the trade name of "Nyflexmet."

This hose is of unusual construction, being built of spirally twisted ribbon-like strips of pliable steel or copper. The edges are formed into interlocking lips, the joints of which are packed with asbestos or rubber. The result is that the hose, while obviously very strong, is yet flexible and capable of standing high pressures. The hose may be lined and also covered with lead, which is pressed into the spiral grooves, and forms an intimate metallic union between casing, core and body. This lead lining and outer casing protect the main structure from corrosion by moisture or acids, thus making the hose acid proof, with the great advantage of being at the same time flexible. It is adaptable for conveying acids, chemicals and acid liquors which would quickly attack other materials. This hose may be used in chemical and metallurgical works, refineries, paper mills, gas plants and dye works, and for conveying liquids under heavy pressures. Foaming and viscous alkalies are claimed not to injure it.

FLEXIBLE LEAD-LINED HOSE.

Another important application may be found in carrying ammonia and anhydride under the high pressures used in refrigerating machinery, etc. It is claimed that owing to the construction of the hose it will stand rough usage, and the freezing of any liquid in the hose will not cause it to crack.

The adjoining illustration shows a section, the method of construction, lining and covering. This hose is manufactured in all sizes from ⅛ inch up to 12 inches internal diameter and any desired length. The couplings used correspond with the regular United States standard fittings, and are claimed to make a joint as strong as the hose itself.

The construction enables its use as a self-supporting pipe line, where supports or uprights cannot be placed near together, such as for crossing wide roads, chasms, etc., and in mines. It is claimed that a pipe 2 inches in diameter can be stretched 300 feet without support. The "Nyflexmet" hose is manufactured by the New York Flexible Metallic Hose & Tubing Co., 170 Lafayette Street, New York City.

Cold Galvanizing.

Galvanizing or coating with zinc is still carried out in this country mostly by the hot process by dipping or wiping, but in recent years a strong tendency has become manifest in favor of the "cold process," that is, zinc plating by electrolysis. The reasons are twofold. First, the superior qualities of electrolytic zinc coatings are now better understood. Second, the cost of the electrolytic process has been very largely reduced by the invention of suitable special devices.

Galvanizing by the hot process has the disadvantage that the coating is uneven and that therefore exaggerated quantities of zinc are required if the coating shall have at any point a certain maximum thickness. It is true that by the introduction of the wiping process the amount of zinc required has been reduced, but this has been done at the expense of the quality of the deposit. While the finished coatings certainly look well, yet in practice they are not found to be a protection against corrosion. This simply means that coatings obtained by the hot process must have a certain thickness if they shall be effective.

It is all wrong to believe that any zinc coating on an iron surface will afford protection, however it is applied. The looks of a zinc coating do not prove anything. If one were to study with the eye the quality of the coating it would be necessary to observe with the microscope the connection between the iron and the zinc. The quality of the coating depends essentially on the molecular joint or weld, so to speak, between the iron and the zinc, and it is for this reason that the method by which the zinc is put onto the iron is of enormous importance, as has been repeatedly pointed out in our columns.

We may refer especially to the extended researches of Prof. Charles F. Burgess, who proved conclusively a great superiority of electrolytic coatings over hot-process coatings with respect to adherence of the coating to the iron, its toughness and strength and resistance against chemical effects from the atmosphere. Prof. Burgess' results have been fully confirmed by independent investigators in Europe, so that there can be no longer any doubt that an electrolytic coating, even when thinner, affords better protection than a coating obtained by the hot process. A further advantage is that with electrolysis it is easy to obtain a uniform coating all over the surface if only the arrangements are properly made, and by adjustment of the current and of the time of electrolysis the coating may be made of any desired thickness.

The chief drawback which has held back the progress of the cold galvanizing process on a large commercial scale has been the prevailing idea that the process is too expensive. Now, as a matter of fact, if the process is carried out in a way in which

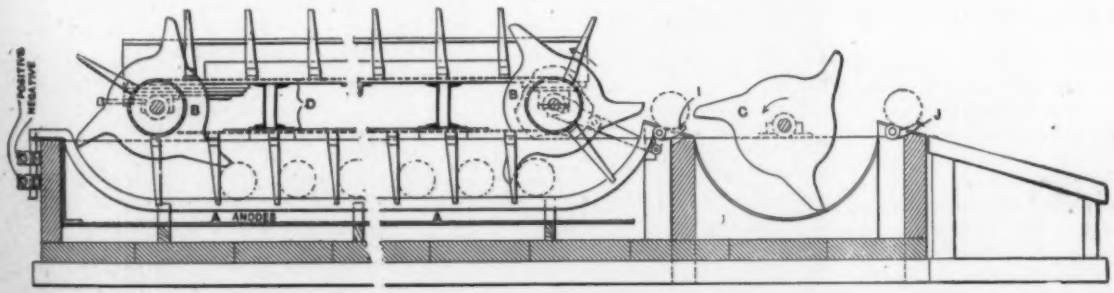


FIG. I.—AUTOMATIC BAR IRON AND PIPE GALVANIZING APPARATUS.

some of the old- fashioned electroplaters still do their work, the cost is high. But this is not the fault of the process but the fault of the people who apply old-fashioned manufacturing methods to a modern process.

By introducing modern manufacturing methods—that is, essentially by devising special labor-saving machinery—the cost of cold galvanizing can be largely reduced and has been reduced. It is clear that the new process dispenses with the employment of the dross and skinning men, and there is no attendant required to keep the pots hot all through the 24 hours. On the other hand, it is evident that to obtain the highest economy with the new process new devices of handling different materials had to be invented, and these devices must differ in their construction according to the character of the material to be treated.

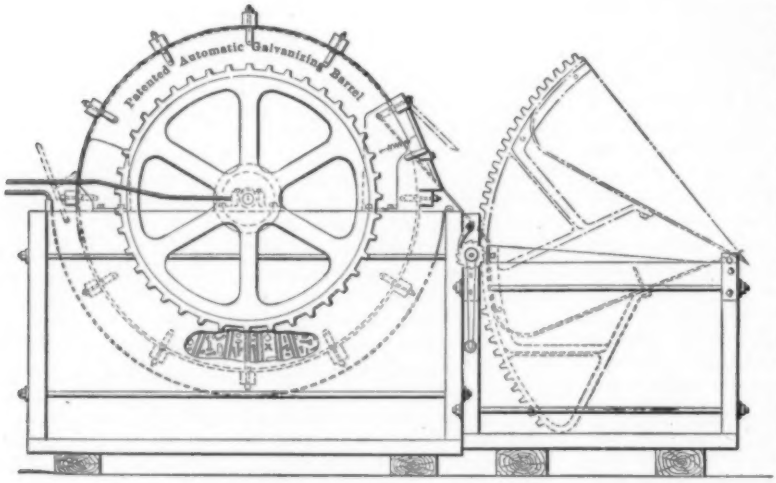


FIG. 2.—BARREL FOR GALVANIZING SMALL ARTICLES.

In this field the United Electro Galvanizing Co., of Brooklyn, N. Y., and its president, Mr. Louis Potthoff, have done important pioneer work. In the following some devices invented by Mr. Potthoff and now employed on a large scale are described:

Bar-Iron and Pipe-Galvanizing Apparatus.

The apparatus is suitable for the treatment of all long material from 6 inches up to bar-iron or pipes 20 feet or longer. Pipe can be coated inside and outside or outside only and sheet metal may also be coated in the same apparatus. The advantages claimed for this apparatus are that the coating is perfectly smooth, that the cost is less than half that of hot galvanizing, and that no experienced labor is required, since the apparatus works perfectly automatically and continuously.

Fig. 1 is a longitudinal section of the apparatus, showing two wooden tanks; the larger one, at the left, containing the galvanizing solution, and the smaller one, at the right, containing hot wash water. Two endless chains D are provided with which are connected pins of non-conducting material. The three-arm cast iron carriers B at the two ends of the tank introduce and discharge the pipes (indicated by dotted lines), and the pins push the pipes along through the tank on two copper bars connected to the negative pole, so that the pipes form the cathodes and are plated with zinc. These runways in the galvanizing tank slant so that the point of contact is constantly changing and an even coating is deposited on the pipes. The anodes are indicated by A.

The operation is obviously continuous. The tank is constantly kept filled, and while one pipe or bar drops out on the right-hand another one enters on the left-hand. By increasing or reducing the speed the thickness of the coating is increased or reduced. The thickness of the coating may also be increased if the length of the tank is increased. When a pipe arrives at the right end it pauses for a moment just before being finally discharged. It then passes over into the wash tank filled with hot water and provided with a three-arm carrier C, which pushes the pipes finally over to the place J, from where it is discharged out of the apparatus. The capacity of one tank, as constructed by the United States Electro Galvanizing Co., ranges from 3 to 6 tons per 10 hours.

Barrel for Plating Small Articles.

The apparatus shown in Fig. 2 is suitable for galvanizing all classes of small material, such as nails, screws, bolts, nuts, small castings, stampings, etc. The cut shows the latest improved form patented by Mr. Potthoff. The galvanizing barrel

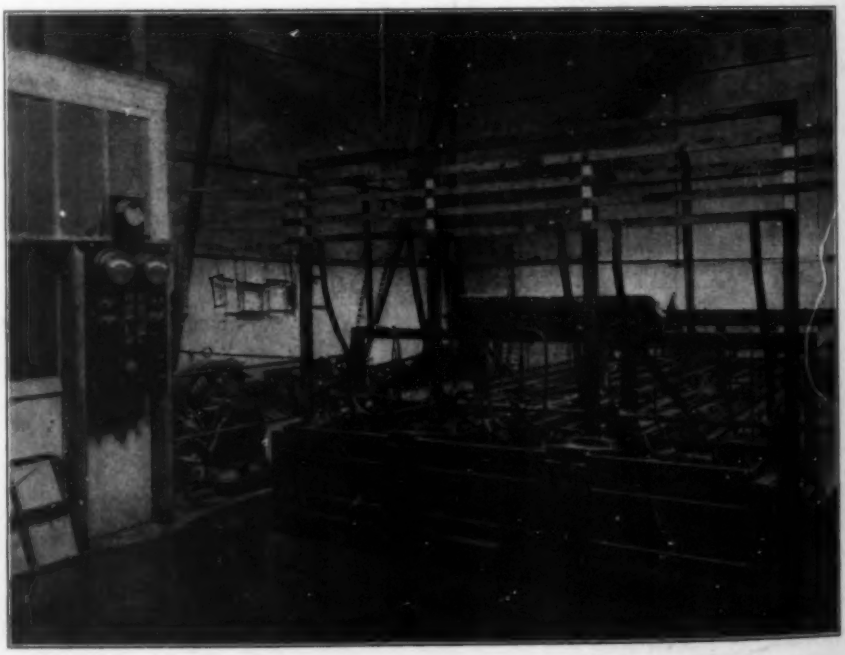


FIG. 3.—INTERIOR OF COLD GALVANIZING PLANT.

is again shown at the left and the washing tank at the right, the small articles being charged into the barrel at the left, and being discharged into the washing tank through the door

ordinarily closed by springs clearly shown in the illustration. While the material is in the tank it is thoroughly agitated so that a uniform coating on all parts of the little articles is obtained. The wash tank may be raised and lowered according to circumstances so as to keep the material sufficient time in the water and finally to discharge it. Mr. Potthoff states that the cost of galvanizing is from 30 to 50 cents per hundred pounds. The thickness of coating can be regulated at will. Bolts do not require rethreading. No skilled labor is required, since on account of the automatic action of the barrel the coating is smooth, bright and perfect and does not depend on the operator. The capacity of the barrel built by the United States Electro Galvanizing Co. is from 150 to 200 pounds to one filling, the output per day being 2,000 to 3,000 pounds.

Figs. 3 and 4 give interior views of an English cold-galvanizing plant in which automatic appliances similar to those described above are employed.

Hard Lead Exhaust Fans.

The use of exhaust fans for removing acid gases and for conveying gases in chemical works and metallurgical plants are numerous.

Where gases are encountered which would corrode iron and steel it has been customary to line the body of the fan with an enamel or glass, but still more often to use a stoneware fan.

The Schutte & Koerting Co., of Philadelphia, are now constructing fans built entirely of hard lead with the exception of the shaft, which is lead covered.

This material is very strong and fans constructed of it can be run at high speed.

These hard-lead exhaust fans are finding use in sulphuric acid plants, and may be placed between the glover tower and the first chamber, or between two Gay-Lussac towers where these are used, or at the end of the system, as circumstances may demand. The fan insures continuous draught under all conditions, resulting in economy of operation, the increased production often being 15 to 20 per cent without enlarging the burners.

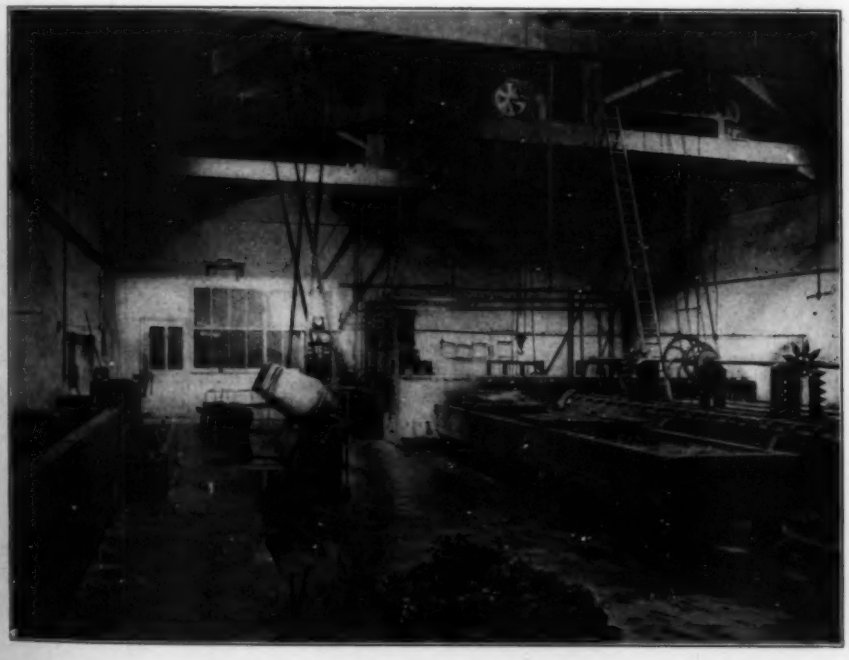
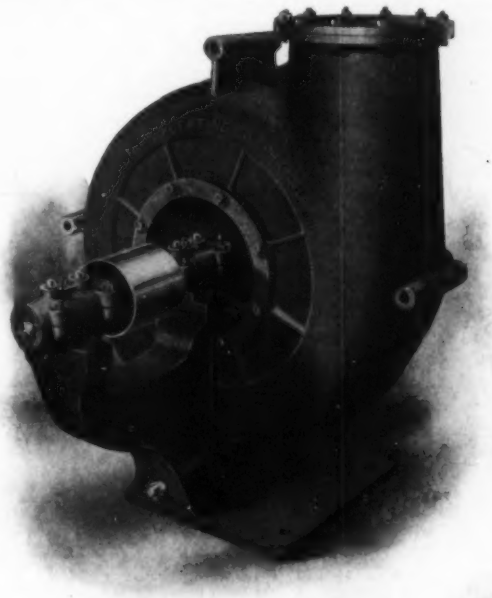


FIG. 4.—INTERIOR OF COLD GALVANIZING PLANT.

The fan may be set to discharge vertically either top or bottom, or horizontally in either direction. When intended to run at low speeds the bearings are massive and strong, being provided with self-oiling devices. The standard type is arranged for belt drive but electric drive may be substituted if desirable.



EXHAUST FAN.

The adjoining illustration shows a fan of this type which the Schutte & Koerting Co. build in eight sizes, ranging in output from 730 to 6,800 cubic feet per minute, the horsepower required varying from .28 to 3.5.

Some idea of the work these fans will accomplish can be gathered from the fact that the largest size has a capacity of 6,800 cubic feet. This fan is operated at 450 r. p. m., the diameter of the inlet is 28 inches, and it is driven by a pulley 16 inches in diameter by 8 inches wide.

Sand-Lime Brick.—The manufacture of sand-lime brick is rapidly becoming an important industry in the United States and is likely to be as profitable here as it has been in Germany for the last ten years. A sand-lime brick is essentially a mass of sand cemented by hydrous lime silicates. The bricks can easily be made with a crushing strength of over 4,000 pounds per square inch and a tensile strength of over 200 pounds per square inch. A brick plant for making sand-lime bricks, near Sayreton, Ala., is briefly described in a paper by Charles Butts in the annual economic bulletin (No. 315) of the United States Geological Survey for the year 1906.

Automatic Still.

The adjoining illustrations show the construction of the automatic still built by the F. J. Stokes Machine Co., of Philadelphia. The raw water enters the lower end of the condenser, and in condensing the distilled water is heated to the boiling point by the time it reaches the overflow. A port connecting the still and condenser feeds this boiling water to the still and maintains a constant level, the action being therefore continuous.

The distinctive feature of this still is the manner in which the steam generated in the still is conducted down through the

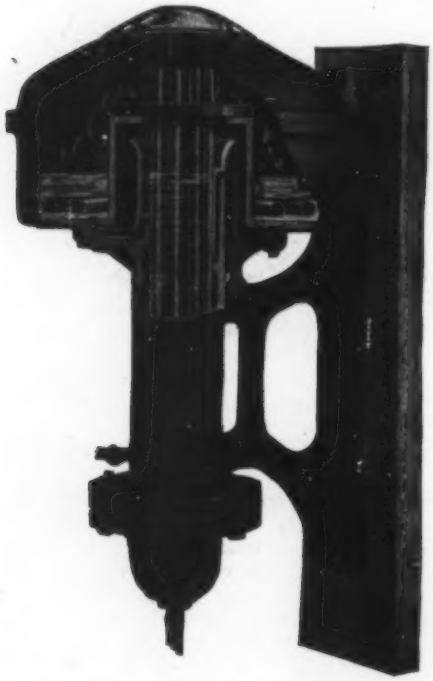


FIG. 1.—SECTIONAL VIEW OF STILL.

condenser, raising the temperature of the feed water to such a point that the ammonia gas is released and driven off. A further advantage of this preliminary boiling is that very little additional heat is required to vaporize it.

Either live steam or gas may be used for heating, and as the steam and water systems are entirely distinct, no oil from steam ever goes into the distilled water. All exposed iron surfaces are tinned to prevent corrosion.

An animal charcoal filter, through which the distilled water percolates on leaving the still, effectively removes all odor and gives the water sparkle and life. The still is made in four sizes for the treatment of 5, 10, 25 and 60 gallons per hour, respectively. The largest size is built in sections, and several stills of this size can be connected in series, thus increasing the total capacity to 300 gallons per hour for five stills in series.

Actual tests of the still of a capacity of 10 gallons per hour have shown that using steam equivalent to 100 pounds at a pressure of 15 pounds, and 125 gallons of condensing water, 10 gallons of distilled water were produced in 1 hour, showing that only 12 gallons of water are required for condensing, which is a very low value.

Notes.

Concentration Mills and Machinery.—A catalog of 92 pages with this title has recently been issued by the Colorado Iron Works Company, of Denver. The book is introduced by a very well written chapter on the scope and fundamental principles of concentration, with a list of specific gravities. Then follow illustrated description of a great many types of crushing, pumping and concentrating machinery.

Cleaning Castings.—The increasing application of the method of cleaning castings by the sand-blast method is indicated by the fact that during the past eight months the Thomas W. Pangborn Co., foundry equipment specialists, have installed their sand-blast machinery in 88 plants, including grey iron shops, malleable shops, steel shops, brass shops and aluminium shops.

New Applications of Gas Producers.—The Industrial Gas Co., New York City, has recently contracted to furnish a 600-hp. Harvey anthracite gas producer to the Albany Chemical Co., where it will be used principally for heating rotary vats in connection with the preparation of chemicals, and also to operate a small gas engine. The equipment will include an updraft generator, an economizer for reheating air and generating steam for gas making, a wet scrubber for cooling

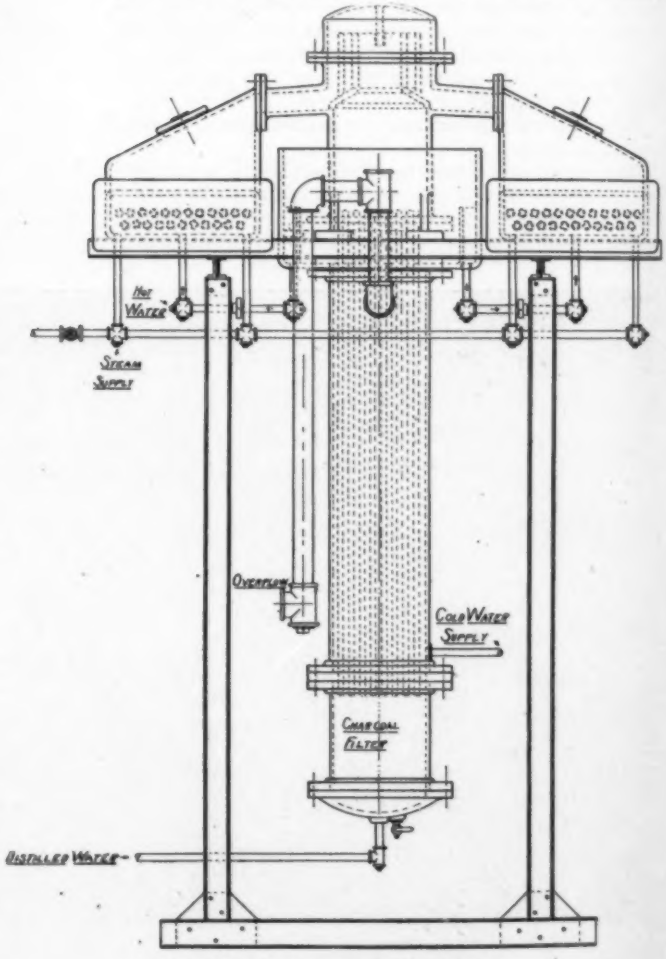


FIG. 2.—DIAGRAM OF AUTOMATIC STILL.

and cleaning the gas and a dry scrubber for drying it. This installation is principally interesting because it is the first in connection with this class of work in the chemical industries. The same company will install eight 12-ft. Herrick generators

for raw bituminous producer gas, to be used in connection with glass melting, at the works of the American Window Glass Co., Hartford City, Ind., eight for Jeanette, Pa. and thirteen for Arnold, Pa.

Electric Motors in a Rolling Mill.—The now celebrated order for 150,000 tons of steel rails placed by the Harriman lines with the Tennessee Coal & Iron Co., for delivery after March, 1908, has forced the latter company to extend its works, in order to handle its greatly increased business. The Tennessee Coal & Iron Co. has just placed an order with the Crocker-Wheeler Co., of Ampere, N. J., for the complete electric motor equipment of its new steel rail mill at Birmingham, Ala. The order includes fifteen Crocker-Wheeler form W rolling mill motors. The line of form W motors is designed for the ardous service of rolling mills, and has attracted very favorable attention in the steel world for its ruggedness and simplicity of design. The order aggregates about 575 hp.

Fire-Brick and Special Refractories.—The consolidation is announced of the Christy Fire-Clay Co. and the Laclede Fire-Brick Co., under the firm name of the Laclede-Christy Clay Products Co. Two plants are controlled and operated by this consolidation—the Christy plant and the Laclede—both situated at St. Louis, where the offices of the company will continue to be. The new company will continue to manufacture as heretofore the high-grade Christy and Laclede products. These include the standard fire-brick shapes of the various Laclede and Christy grades; silica brick in standard shapes and specials, linings for lime kilns, cupolas, brass furnaces, heating furnaces, rotary cement kilns, etc.; gas house refractories, sewer pipe and fittings, flue and chimney linings and tops, as well as special refractory materials to suit any requirements. The Laclede-Christy Co. are also engineers for gas houses, glass factories, coke ovens, plants and metallurgical works. Their products include a full line of fireclays and all clay products. The officers of the company are: William C. Morris, president; John L. Green, vice-president; Richard D. Hatton, secretary and treasurer. The superintendent of the Christy plant is J. A. W. Schoedel, and of the Laclede plant Frederick Talbot.

Electric Steel Furnace.—We have received from the American Electric Furnace Co.—which are the sole owners of the United States and Canadian rights to manufacture, lease or sell electric reduction furnaces under the patents of Colby, Kjellin and others—their well-illustrated pamphlet on the induction furnace. The principle of construction and operation is discussed and the general working arrangement of an induction furnace plant is given. A comparison with other methods is then made, especially with the crucible process and the open-hearth process. We use this opportunity to correct a misprint in the address of this company given in our last issue. The correct address is 45 Wall Street, New York City.

Power.—The American Gas & Electric Co., of Philadelphia, Pa., own and operate central station light and power plants in ten cities. They have special facilities and are enabled to quote attractive prices for power to electrochemical plants. They are backed by the General Electric Co., and their power plants are situated in the following cities: Atlantic City, N. J.; Scranton, Pa.; Altoona, Pa.; Marion, Ind.; Munsey, Ind.; Auburn, N. Y.; Conshohocken, Pa.; Canton, Ohio; Wheeling, W. Va.; Bridgeport, Ohio.; Rockford, Ill.

Electric Measuring Instruments.—We have received from the Leeds & Northrup Co., of Philadelphia, their exceedingly interesting and well illustrated 1907 catalog on electrical measuring instruments. After a brief review of the manufacturing facilities of the company, descriptions of the various electrical instruments made by them are given, and it should be acknowledged that, contrary to custom in trade catalogs, the descriptions in this one are quite specific in all cases, so that those who have occasion to consult the catalog may get definite ideas of the apparatus. Very considerable attention is also given to the illustration and description of details. Among the new instruments described for the first time in this catalog we notice a line of resistance thermometers. Since this is a subject of great interest to our readers, we will describe them at greater length in our next issue.

Maximum Results from Fuel.—A pamphlet has been issued by Prof. William Jones, chemist of the Metropolitan Street Railway Co., of New York City, on the chief requirements for obtaining the maximum results from fuel. Not only must the temperature in the flue be low, while that of the furnace is high, but the volume of the flue gas must be as small as possible. To attain this result it is necessary to allow only so much air to enter the furnace as is necessary for complete combustion of the fuel, that is the flue gas must not have any large excess of unused air, which excess is shown by the percentage of CO_2 in the flue gas. A diagram is given, showing at a glance the loss of fuel in flue gas as a function of the temperature and of the percentage of CO_2 in the gas. Prof. W. Jones has invented a gas analyzer which indicates continuously the amount of carbonic acid gas in the flue gas as it leaves the furnace.

Pyrometer.—We have received from Mr. Charles Engelhard, 41 Cortlandt Street, New York City, their latest pamphlet on the well-known Heraeus-Le Chatelier pyrometer. A brief description of the instrument is followed by a long list of plants in which it is in use in this country. Various useful tables of melting points, etc., are given at the end of the pamphlet.

Pyrometer.—One of the best and most clearly written discussions of the principles of radiation pyrometry may be found in the April issue of the *School of Mines Quarterly*, in an article by Messrs. C. H. Wilson and F. Maeulen on the Féry radiation pyrometer. The description is so clear that it should be understood by anybody who is interested in the subject at all. The paper has been reprinted and copies of the reprints may be had from the Wilson-Maeulen Co., 120 Liberty Street, New York City, for the asking.

Forged Steel Flanges.—We have received from the American Spiral Pipe Works, of Chicago, their 1907 flange catalog, containing a large amount of general and detailed information on flanges for power work. The catalog, which covers almost 100 pages, is profusely and exceedingly well illustrated.

Flange Union.—In endeavoring to devise a flange union which can always be made tight and remain so without the use of a gasket, the Western Tube Company, of Kewanee, Ill., has developed the "Kewanee" flange union which is made entirely of the highest grade malleable iron, excepting the brass seat, which is made practically integral with the malleable part by the only method which experience has shown to be enduring, by screwing in. There is no danger of breaking these malleable flanges in putting the joints together. The convex surface of malleable comes in contact with the concave surface of brass; this junction of hard and soft metal insures a more perfect seat than could be obtained otherwise and it makes no difference whether the alignment is straight or not.

Electrolytic Sterilization of Water in Swimming Baths.—Dr. Samuel Rideal, of London, has been permitted by the Westminster City Council, under protective conditions, to carry out experiments on the sterilization of water at one of the swimming baths by electrolytic chlorine, with the object of ascertaining whether the water need be changed as frequently when kept purified as at present.

Mining Machinery.—The John A. Traylor Machinery Co., of Denver, Col., have sent us their illustrated bulletins F and G, dealing with pumping machinery and with laboratory and sampling equipment, respectively. Bulletin F contains, besides illustrations of pumps, etc., some useful tables on the horse-

power required, the capacity and the cost of operating electrically-driven centrifugal pumps.

Gen. William J. Palmer has given the Engineering School of Colorado College, Colorado Springs, the sum of $12,000, to be expended immediately upon additional equipment of the engineering laboratories for senior work.

The Westinghouse Electric Works at East Pittsburg have established another new record by shipping during the month of May no less than 750 carloads of electrical machinery, or an average of thirty carloads a day, aggregating 10,000 tons, and representing in value about $4,000,000.

The Allis-Chalmers Co. states that the recent cry of dull times, raised by the various manufacturing interests of the country, finds no echo in their own business company which continues to show a steady gain. During the month of May this company shipped from its works 553 cars of machinery, which was a gain of 20 cars over the record established for April. In April the aggregate weight of shipments was 21,680,847 pounds, while for the months of May the figure has risen to 23,772,242 pounds, making a total by weight for the two months of 45,453,089 pounds. Cars bearing this enormous quantity of machinery, if coupled in one train, would have covered a distance of about eight miles.

Paint.—Voltax liquid compound, which for the past year has been coming into extensive use as a weatherproof paint, has been specified by the Board of Education, of New York City, as a standard paint in Sec. 70, relating to painting and enameling on screens, radiators, coils and connections. Voltax will be used as a paint on pneumatic pipes, valves, canvas coverings on smoke pipes, grease extractors, economizer connections, feed water heaters, receiving tanks, boiler and incinerator fronts and on all fixtures, boilers, engines, pumps, blowers, foundations, steel bands and tanks set in the floors. To meet the growing demands for the compound the Electric Cable Co., of 17 Battery Place, New York City, is now erecting an extensive addition to their plant at Bridgeport, Conn.

The Electric Cable Co., of New York, has just bought the entire business of the Eastern Wire & Cable Co., of Roxbury, Mass. The latter company is one of the oldest and largest manufacturers of rubber-covered wires and cables in New England. Its entire equipment will be removed to Bridgeport, Conn., where it will be installed in the plant of the Electric Cable Co., which has in course of construction a large addition to its plant.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

CALCIUM CARBID (*Continued.*)

No. 571,084, Nov. 10, 1896, Hilliary Eldridge, Daniel Johnson Clark and Mahlon W. Wambaugh, of Galveston, Tex.

Produces calcium sodic carbid containing a trace of borax by subjecting a mixture of pulverized quicklime, 72 bushels; carbon, 40 bushels; soda, 4 bushels, and borax, ⅛ bushel, to the fusing heat of an electric furnace. Common washing soda may be used. Both the carbonate of soda and borax should be preliminarily heated to expel water of crystallization; they are said to have an action on the lime "similar to a flux or an acid," greatly accelerating the reduction of the lime.

No. 572,636, Dec. 8, 1896, James Ellicott Hewes, of Philadelphia, Pa.

Produces carbid in a furnace having an inclined hearth of fire-brick. A hinged metal door, serving as the cathode, swings against the lower end of the hearth and closes the bottom of the angular chamber. The anode is a carbon slab sliding on the hearth and automatically fed. The product, smelted by an electric arc, is discharged through the opened door into a chamber beneath. The dust is drawn off through a lateral suction pipe, and the "ash, slag" and unreduced material passed downward through a screen to a conveyor.

No. 578,685, March 9, 1897, Edwin R. Whitney, of Manchester, N. H.

Springs crossing arcs between the ends of four radial horizontal electrodes consisting of powdered charcoal, forced through and out of tubes which terminate at the walls of the smelting chamber. A mixture of lime and carbon is fed downward through the arcs. The powdered charcoal is fed forward and caused to cohere by screw conveyors, but a binder of tar or syrup may be added. The liquid calcium carbid runs out of the smelting chamber through a grating at the bottom. The crossed arcs concentrate the heat, and each arc counteracts the tendency of the other one to blow aside the particles of lime and carbon.

No. 583,498, June 1, 1897, James T. Morehead, of Spray, N. C.

Builds up an ingot of carbid in a rectangular smelting chamber, three walls of which consist of brick. The front wall consists of removable superposed sheets of iron, sliding in grooves. The lower electrode is a layer of carbon supported on a metal base plate. The upper electrode is an adjustable depending carbon rod. In starting the furnace, the lower iron sheet only, is put in place, and the enclosure is filled with pulverized lime 20 parts, and carbon 12 parts. When the charge has been converted into carbid the front wall is raised by putting another iron sheet in place, more material is charged in and reduction proceeds. When the furnace is filled with carbid, the front wall is removed and the unreduced charge around the ingot falls away and is used, while hot, as a new charge. The removal of the wall facilitates the cooling and removal of the carbid by grapples. By keeping the lower end of the upper electrode near the upper edge of the front wall the gases readily escape, without striking the electrode, prolonging its life.

No. 587,138, July 27, 1897, Isaiah L. Roberts, of Niagara Falls, N. Y.

Produces a continuous slab of carbid on a moving horizontal endless belt of wire cloth, each wire being covered with asbestos thread. Two horizontal electrodes passing through ball and socket joints, are arranged to spring an arc above the belt. The belt and electrodes are surrounded by a sheet iron casing, having an upper supply hopper at one end of the belt and a flue at the other end. As the arc effects reduction within the charge lying on the belt, the electrodes are gradually moved apart until parallel, the current passing through the slab of carbid, acting as a resistance. As the slab passes off the end of the belt it is gradually broken off in pieces, and the unreduced material falls into a receptacle and is returned to the hopper. The electrodes may be initially parallel and a current passed between them through pieces of broken carbon, constituting a temporary conductor until the bridge of carbid is formed. The process may be used for the production of carborundum or reduction of metallic oxids.

No. 587,509, Aug. 3, 1897, Isaiah L. Roberts, of Brooklyn, N. Y.

Produces a slab of carbid on an endless horizontal belt, an electric arc being sprung between the ends of transverse carbon rods, arranged over the belt. The belt and electrodes are arranged in a metal casing, provided with superposed supply hoppers at one end. The scraper shapes the charge as the belt carries it forward to the arc. An electromagnet deflects the arc downward onto the charge, producing flakes or scales of carbid, which are scooped off by a blade at the discharge end of the belt. The unreduced mixture falls onto a belt conveyor and is returned to the lower hopper. The belts are preferably of asbestos cloth. The whole chamber may be filled with a non-oxidizing atmosphere, for example, carbonic oxid or hydrogen.

NEW BOOKS.

SCIENTIFIC PAPERS. By Josiah Willard Gibbs. 2 volumes. 434 pages. Vol. I., $5.00; Vol. II., $4.00. New York City: Longmans, Green & Co.

BLAST FURNACE CALCULATIONS AND TABLES FOR FURNACE MANAGERS AND ENGINEERS. Containing rules and formulas for finding the dimensions and output capacity of any furnace, as well as the regular outfit of stoves, heating surface, volume of air, tuyere area, etc., per ton of iron per day of 24 hours. By John L. Stevenson. 44 pages. Price, $2.00. New York: D. Von Nostrand Co.

OPEN-HEARTH STEEL CASTINGS. By W. M. Carr. 118 pages; 19 illustrations. Price, $1.50. Cleveland, Ohio: The Penton Publishing Co.

PRACTICAL METAL TURNING. A handbook for engineers, technical students and amateurs (reissue of "Engineers' Turning"). By J. G. Horner. 404 pages; 485 illustrations. Price, $3.50. New York City: Norman W. Henley Publishing Co.

GEM CUTTER'S CRAFT. By Leopold Claremont. 296 pages. Price, $5.00. New York City: The Macmillan Co.

REPORT ON PROGRESS OF INVESTIGATIONS OF MINERAL RESOURCES OF ALASKA IN 1906 (United States Geological Survey Bulletin No. 314). By Alfred Hulse Brooks and others. 235 pages; illustrated; paper binding. 30 cents. Washington, D. C.: Superintendent of Documents.

PRACTICAL METHODS OF INORGANIC CHEMISTRY. By F. M. Perkin. Price, $1.00. New York City: D. Van Nostrand Co.

COMPRESSIBILITIES OF THE ELEMENTS AND THEIR PERIODIC RELATIONS (Carnegie Institute Publications No. 76). By Theodore W. Richards. 67 pages; 8 illustrations. Price, 50 cents. Carnegie Institute.

FURTHER RESEARCHES CONCERNING THE ATOMIC WEIGHTS OF POTASSIUM, SILVER, CHLORINE, BROMINE, NITROGEN AND SULPHUR. By Theodore W. Richards, A. Staehler, G. S. Forbes, E. Mueller and G. Jones (Carnegie Institute Publications No. 69). 88 pages; 4 illustrations. Price, 50 cents. Carnegie Institute.

INDUSTRIAL ALCOHOL. A practical manual on the production and use of alcohol for industrial purposes and for use as a heating agent, as an illuminant and as a source of motive power. By J. G. McIntosh. 252 pages; illustrated. Price, $3.00. New York City: D. Van Nostrand Co.

ON THE ADSORPTION OF WATER VAPOR OF CERTAIN SALTS IN AQUEOUS SOLUTION BY QUARTZ. By Lyman James Briggs. 26 pages; illustrated. Price, 25 cents. Ithaca, N. Y.: *Journal of Physical Chemistry.*

PRACTICAL PAPER MAKING. A pamphlet for paper makers and owners and managers of paper mills, to which are appended useful tables, calculations, data, etc., with illustrations reproduced from Micro-Photographs. By George Clapperton. 226 pages. Price, $2.50. New York: B. Van Nostrand & Co.

DIFFERENT METHODS OF PURIFYING WATER. Read before the Engineers' Club of Philadelphia, Nov. 17, 1906. By J. A. P. Maignen. 67 pages; illustrated. Authorized reprint from Vol. XXIV, No. 1 (January, 1907), of the copyrighted proceedings of the Engineers' Club of Philadelphia.

HANDBOOK ON ENGINEERING. The practical care and management of dynamos, motors, boilers, engines, pumps, inspirators and injectors, refrigerating machinery, hydraulic elevators, electric elevators, air compressors, rope transmission and all branches of steam engineering. Sixth edition revised and enlarged. 1,072 pages; illustrated by tables and diagrams. Bound in leather. Price, $3.50. St. Louis:, H. C .Tulley & Co.

LABORATORY AND FACTORY TESTS IN ELECTRICAL ENGINEERING. By G. F. Sever and F. Townsend. Second edition revised and entirely rewritten; illustrated. Price, $2.50. New York City: D. Van Nostrand Co.

MODERN POLYPHASE MACHINERY. By Andrew Stewart. 296 pages; 181 illustrations. Price, $2.00. New York: D. Van Nostrand Co.

BOOK REVIEWS.

COMMERCIAL ORGANIC ANALYSIS. Vol. II., Part 3: Phenols, Aromatic Acids, Resins, Essential Oils. By Alfred H. Allen, F. I. C., F. C. S. Third edition, rewritten and revised by the author and A. R. Tanhard, F. C. S. 8vo. pp. xi. + 547. Price, $5.00 net. Philadelphia: P. Blakiston's Son & Co.

This monumental work on organic analysis consists in reality of eight volumes, two of which have reached their second edition and five their third edition. The usefulness of these volumes to organic chemists can hardly be over-estimated. They give the properties, methods of valuing and proximate analytical examination of the various organic chemicals and products in practical use, together with the detection and determination of their impurities, adulterations and products of decomposition. The third edition of this particular volume is, moreover, not a mere reprint, for the chapters on resins and essential oils, forming nearly three-quarters of the book, are almost entirely new, occupying nearly ten times the space devoted to them in the previous editions. The ground is covered so exhaustively and the treatment is so clear and masterly, that this work must certainly become a *vade mecum* for everybody working with these organic compounds.

ALTERNATING-CURRENT MOTORS. By A. S. McAllister, Ph. D. 278 pages; 122 illustrations. Price, $3.00. New York: McGraw Publishing Co.

This is the first book ever published on the theory of alternating-current motors, including the single-phase commutator motors which are now becoming of such importance in electric traction schemes for main railways. There are sixteen chapters, dealing respectively with single-phase and polyphase circuits; outline of induction motor phenomena; observed performance of induction motor; induction motors as frequency changes; Heyland induction motor; single-phase induction motor; graphical treatment of induction motor phenomena; induction motors as asynchronous generators; transformer features of the induction motor; magnetic field in induction motors; synchronous motors and converters; electromagnetic torque; simplified treatment of single-phase commutator motors; motors of the repulsion type treated both graphically and algebraically; motors of the series type treated both graphically and algebraically; prevention of sparking in single-phase commutator motors.

Much attention is given to graphical diagrams and to examination of the facts upon which they are based. Mathematics is employed "merely as a short-hand method of stating facts." There is much truth in the following statement, which holds good for certain physico-chemical theories as well as for the analysis of alternating-current motors: "Beyond any doubt much confusion is created in the mind of an engineering student when an attempt is made to express certain *assumed* relations in the form of mathematical equations, and then to transform the equations to obtain a result which he is asked to believe expresses physical *facts*, merely because the equations, considered mathematically, have been properly transformed."

The theory of alternating-current motors appears at first sight so very far away from anything which might interest electrochemists that the one point where both fields come in touch should be pointed out here. This is in the theory of the electric induction furnace which has now become a commercial apparatus for making high-grade steel and which will become, presumably. even more important for brass smelting. The induction furnace is a special case of a transformer. The induction motor is a special case of the general alternating-current transformer (with movable secondary). The theory of the induction motor is essentially based on that of the transformer; and many of the results can be taken over directly into the theory of the induction furnace. The induction motor and the induction furnace have this feature in common, which

STANDARD BOOKS

METALLURGY

ORE DRESSING
By ROBERT H. RICHARDS
Of the Massachusetts Institute of Technology
Two volumes, 6½ x 9½, cloth, 1250 pages, profusely illustrated, per set $10.00.

MODERN COPPER SMELTING
By EDWARD DYER PETERS
600 pages 6½ x 9½, cloth, $5.00.

PRINCIPLES OF COPPER SMELTING
By EDWARD DYER PETERS
Professor of Metallurgy, Harvard University
612 pages 6½ x 9½, cloth, $5.00.

PYRITE SMELTING
By T. A. RICKARD
210 pages 6½ x 9½, $2.00.

THE METALLURGY OF LEAD
By H. O. HOFMAN
Professor of Metallurgy, Massachusetts Institute of Technology
600 pages 6½ x 9½, $6.00.

HYDROMETALLURGY OF SILVER
By OTTOKAR HOFMANN
336 pages 6½ x 9½, cloth, $4.00.

MANUFACTURE AND PROPERTIES OF IRON AND STEEL
HARRY HUSE CAMPBELL
640 pages, 6 x 9, illustrated, $5.00.

METALLURGICAL CALCULATIONS
PART I. CHEMICAL PRINCIPLES
By PROF. JOSEPH W. RICHARDS
200 pages, cloth, $2.00 net.

METALLURGICAL CALCULATIONS
PART II. IRON AND STEEL
By PROF. JOSEPH W. RICHARDS
250 pages, cloth, $2.00 net.

IRON, STEEL AND OTHER ALLOYS
By PROF. HENRY M. HOWE
Second edition, 513 pages, 123 illustrations. Price $5.00 net.

METALLURGICAL LABORATORY NOTES
By PROF. HENRY M. HOWE
154 pages, 44 illustrations. Price, $2.50 net.

In this book the author describes ninety-one laboratory experiments conducted by students in the Department of Metallurgy of the School of Mines of Columbia University, and deals with Pyrometry, Calorimetry, Metallography, Heat Treatment of Steel, etc.

ELECTROMETALLURGY

ELECTRIC SMELTING AND REFINING
By DR. W. BORCHERS. Translated by W. G. McMILLAN.
Second edition, 562 pages, 255 illustrations. Price, $7.00.

This book treats of all those metals in the extraction and working of which the electric current has found any application. A survey of the purely metallurgical methods of treatment has been added to each chapter, so the two methods may be readily compared.

THE ELECTRIC FURNACE
By HENRI MOISSAN
Authorized Translation By DR. VICTOR LENHER
305 pages, illustrated. Price, $2.50.

A complete description of this recent apparatus and its application to the fusion and volatilization of refractory substances. The work is brought up to date by accounts of recent development.

Any of the above books sent postpaid on receipt of price.

We can promptly furnish any Metallurgical, Chemical or Engineering book.

ELECTROCHEMICAL PUB. CO.
Book Dept. **114 Liberty St, New York**

distinguishes them from a well designed ordinary stationary transformer: the large magnetic leakage. To the future theorists of the electric induction furnace Dr. McAllister's book is heartily recommended.

An Investigation of the Borides and the Silicides. By Oliver Patterson Watts, Ph. D. 68 pages; illustrated. Price 30 cents. Madison, Wis.: Bulletin of the University of Wisconsin No. 145.

This is an excellent little monograph on the electrometallurgy of the borides and the silicides, and reflects high credit on the institution from which it has been issued—the laboratory of applied electrochemistry of the University of Wisconsin, which is in charge of Prof. C. F. Burgess.

In the first part of the pamphlet a very useful summary is given of the existing literature on silicides and borides. Most of the work in these fields in the past has been done in France and in Germany. The results of the work of the different experimenters are worked up by Dr. Watts in form of various tables giving the references to original articles, an indication of the method of preparation of the different compounds and notes on their physical and chemical properties. The lists include 23 borides and 36 silicides. It appears that the usual method of preparing these compounds has been that of direct synthesis from the elements; occasionally the metallic oxide has been reduced by boron or silicon, and still more rarely the oxygen compounds of both the metal and the non-metal have been simultaneously reduced by carbon or magnesium.

The second part of the bulletin is an account of the author's own experimental work, which was undertaken for two purposes. The first was to investigate the possibility of preparing the borides and silicides from their commercial available oxygen compounds by a single reaction in the electric furnace; the second was to attempt to produce a new series of definite chemical compounds—the silico-borides.

Dr. Watts first gives an interesting brief summary of the various reducing agents which are available for the electric furnace. Carbon is, of course, the most important metallurgical reducing agent, and for the electric furnaces it has certain advantages. It is cheap, it is a powerful reducing agent at high temperatures, it is so slightly volatile that it remains in the furnace while the product of its action is a gas and hence escapes as soon as formed. The great disadvantage of carbon is its tendency to form carbides with nearly all metals which are produced in the electric furnace, and also in the presence of silica or borax, to form the carbides of silicon and boron, extremely resistant to chemical reagents and difficult to remove from the product desired. In the reduction of the metallic oxides the amount of carbon in the product may be made very small, and even negligible in some cases, by using an excess of metallic oxide in the charge. In the preparation of borides and silicides from their oxygen compounds, however, this method of removal of carbon from the product is not available.

Aluminium has been found an excellent reducing agent mainly as the result of Dr. Hans Goldschmidt's extensive work in this field. Dr. Watts has tried a modification of the Goldschmidt aluminothermic method. The main feature of the latter is, of course, that no outside supply of power is required. Dr. Watts has found that for certain reactions it is useful to combine the aluminothermic method with an electric furnace treatment, and has developed a method of firing in the electric furnace a charge consisting of powdered aluminium and any other powdered substance or substances that he desired to reduce, and obtained a product free from both carbon and aluminium. Besides applying the method to rutile, boric and tungstic "acids" were similarly reduced, and by the addition of a retarder to check the vigor of reaction, ordinary Goldschmidt charges were fired in the electric furnace without accident. (See also our Vol. III., p. 372.)

Another useful reducing agent is calcium carbide, which reduces the same classes of compounds as are reduced by

aluminium, and is especially advantageous to use for mixtures of an oxide with a chloride or a sulphide. It seems to be settled that calcium alloys are produced in certain reductions, but the exact conditions which cause their formation are not yet established. The products are contaminated by carbon under the same conditions, and possibly to the same extent, as if free carbon were used for reduction.

After giving a few notes on the use of aluminium carbide and silicon carbide as reducing agent, the author speaks of calcium which is now made commercially on a large scale and which is believed by Dr. Watts to be superior to any other general reducing agent, where purity of the product of reduction is of importance. "The metal in granular or powdered form can undoubtedly be used as aluminium now is in the Goldschmidt method of reduction. For this use it will be superior to aluminium for different reductions. For one equivalent of oxygen its heat of oxidation is 10 per cent greater than that of aluminium—a sufficient increase to insure the completion of certain reductions that by aluminium are only partial. Mixtures of oxides with calcium will undoubtedly have a considerably lower kindling point than those with aluminium, a quality which will be a convenience, and will tend to cause more rapid and complete reaction. A disadvantage of calcium as compared with aluminium is its oxidation at ordinary temperatures. Another, and possibly serious objection to calcium as a reducing agent in the electric furnace is to be seen in the strong oxidizing action of fused lime upon silicon, boron, chromium, cobalt, iron, manganese, nickel and titanium in the experiments of Moissan."

A few words are then said on the important subject of slags and the general arrangement of the author's experiments is described. In the first experiments he endeavored to make silicides from sulphide ores. For instance, silicon copper from chalcocite, Cu_2S, and silica in one reaction, or ferro-silicon from ferrous sulphite and silica. The equation of the reduction of silicon copper from Cu_2S with calcium carbide as reducing agent is, for instance, $Cu_2S + SiO_2 + CaC_2 = Cu_2Si + CaS + 2CO$. The general results of this series of experiments are that metallic sulphides and silicates can be simultaneously reduced by calcium carbide or aluminium with the production of a silicon alloy. The total yield, based on both the metal and non-metal in the charge, is little above 50 per cent. The use of carbon or a carbide as reducing agent introduces carbon into the metallic silicides of those metals capable of uniting with carbon. The product can be obtained entirely free from sulphur by the use of lime or aluminium. In the course of these experiments the author found a new silicide of molybdenum which is richer in silicon than Mo_2Si_3, and probably has the formula of $MoSi_2$. The author's attempt to use an analogous method for preparing the borides was unsuccessful as was also his attempt to find a series of silicoborides, although his experiments do not prove that such a series does not exist.

The Bulletin is concluded by some interesting notes on the behavior of the horizontal arc furnace in which the experiments were carried out. The resistance of the arc depends upon four factors. First, on the length of the arc. Second, on the current strength; an increase of current increases the cross-section of the arc, and therefore diminishes the resistance. Thirdly, the resistance of the arc depends on the temperature of the furnace. During the rise in temperature at the beginning of an experiment the arc must be gradually lengthened if the current is to be maintained constant. When a cold charge is fed into a hot furnace the current is diminished. Fourthly, the quantity and nature of vapors within the furnace are also a factor of influence on the resistance of the arc. An increase in the amount of vapors of sodium and of silicon lowers the resistance. Possibly there are vapors which are capable of increasing the resistance. From various data the author concludes that the arc tends to maintain a constant ratio between its own resistance and that of the remainder of

Dreaper

The Chemistry and Physics of Dyeing

Being an Account of the Relations between Fibres and Dyes, the Formation of Lakes, and the General Reactions of Colloids, and their Solution State.

By W. P. DREAPER, F.I.C., F.C.S.

Illustrated by Curves and Numerous Tabulated Results

Octavo. *Cloth, net $3.00*

In the present volume an attempt is made to collect and classify the work which has been brought forward to explain the action of dyeing, mordanting and lake formation. The general text books on dyeing devote little space to this particular side of the question. They deal with the operations of the dye-house, rather than with the principles which seem to govern the actual practice of this branch of the industry. It is advisable that the modern dyer should have some knowledge of the general reactions which give rise to the results obtained in the many processes involved in the dyeing and bleaching of textile fibres. Without some such knowledge it is difficult to appreciate their nature, or be in a position to control their working in a systematic manner.

Catalog of Chemical Books sent free upon request

P. Blakiston's Son & Co.

Publishers

1012 Walnut Street, Philadelphia

Handbook of Mathematics

FOR ENGINEERS AND ENGINEERING STUDENTS

By J. CLAUDEL

Translated and Edited from the Seventh French Edition by OTIS ALLEN KENYON

A complete book on Mathematics
It covers very thoroughly
Every branch from Arithmetic to Calculus

TABLE OF CONTENTS

PART I.—ARITHMETIC. I.—Fundamental Operations on Whole Numbers. II.—Properties of Whole Divisors. III.—Fractions and Decimal Numbers. IV.—Powers and Roots. V.—Ratios, Proportions, Progressions. VI.—Diverse Rules. VII.—Logarithms. PART II.—ALGEBRA. I.—Of the Four Fundamental Algebraic Operations. II.—Equations of the First Degree. III.—Powers and Roots of the Algebraic Quantities. IV.—Equations of the Second Degree. PART III.—GEOMETRY. Plane Geometry. Solid Geometry. Problems in Geometry. PART IV.—TRIGONOMETRY.—Plane. Spherical Trigonometry. PART V.—ANALYTICAL GEOMETRY. PART VI—CALCULUS. I.—Differential Calculus. II.—Integral Calculus. III.—Applications of Integral Calculus.

Cloth, 708 Pages. 422 Illustrations. Price, $3.50 Net.
Sent anywhere postpaid for the price
Send Us Your Inquiries Write For Our Catalogs

McGraw Publishing Company

PUBLISHERS, IMPORTERS AND BOOKSELLERS

114 Liberty Street, New York

DUHEM-BURGESS—*Thermodynamics and Chemistry.* A Non-Mathematical Treatise for Chemists and Students of Chemistry. By P. Duhem, Correspondant de l'Institut de France; Professor of Theoretical Physics at the University of Bordeaux. Authorized Translation by George K. Burgess, Docteur de l'Université de Paris; 8vo, xxi + 445 pages, 140 figures. Cloth, $4.00.

GETMAN—*Laboratory Exercises in Physical Chemistry.* By Frederick H. Getman, Ph. D., Fellow by Courtesy of The Johns Hopkins University, Carnegie Research Assistant. 12mo, viii + 241 pages, 100 figures. Cloth, $2.00 net.

LÖB-LORENZ—*Electrochemistry of Organic Compounds.* By Dr. Walther Löb, Privatdocent in the University of Bonn. Authorized Translation by H. W. F. Lorenz, A.M., Ph.D., 8vo, x + 308 pages, 10 figures. Cloth, $3.00.

WEAVER—*Notes on Military Explosives.* By Erasmus M. Weaver, Lt. Col. Artillery Corps, U. S. Army. 8vo, viii + 311 pages. Cloth, $3.00 net.

JOHN WILEY & SONS
43 & 45 E. 19th St., New York City

Eleventh Edition

Metallurgy of Cast Iron

A complete exposition of the processes involved in its treatment, chemically and physically, from the blast furnace through the foundry to the testing machine, being a practical compilation of original research.

By THOMAS D. WEST

Contents

PART I.—Manufacture and Use of Coke—Properties in Ores—Operations of Blast Furnaces—The Different Brands of Pig-Iron and How to Purchase and Use Them Intelligently (27 chapters). II.—Elements in Cast-Iron and Their Physical Effect—Utility of Chemical Analyses and How to Use Them in Making the Different Mixtures of Irons Used in Making Gray and Chilled Castings (21 chapters). III.—Properties of and Methods for Testing Molten Iron—Phenomena in the Actions of Cooling Metal, etc.—Results of Tests in All Kinds of Irons—Best Methods for Testing (23 chapters). APPENDIX. Selected Tables for Use in Furnace and Foundry Work.

Cloth 638 Pages 153 Illustrations Price, $3.00 net

Sent Anywhere, postpaid, for the Price by the

McGRAW PUBLISHING COMPANY

114 Liberty Street :: New York

the circuit, or an arc of fixed length tends to maintain a constant voltage.

When trouble is experienced with the arc, on account of its being unsteady, the source of trouble is found at the anode, and seems to Dr. Watts to be due to insufficient heat to maintain there a layer of carbon vapor.

Electrode losses are due to three causes, according to Dr. Watts; first, volatilization from the crater, confined to the anode; second, oxidation; and, third, disintegration—particles falling from the electrodes. The first cause is independent of, the second and third are dependent upon, the length of electrode within the furnace. The total loss is diminished about 50 per cent by filling the furnace with illuminating gas; this can diminish only the second and third effects. The losses with 2-inch electrodes were double those with 1½-inch diameter. This points to the use of the smallest possible electrodes. In the whole it will be seen that the little booklet contains much useful information.

* * *

Über Ferrosilicium. By Dr.-Ing. Waldemar Pick. 97 pages; illustrated. Printed by G. Braun in Karlsruhe.

We are obliged to the author for sending us this very interesting book, which apparently has no regular publisher and cannot be obtained through the ordinary channels for buying books. But we are glad to learn that Dr. Pick has in preparation—jointly with Dr. W. Conrad—an enlarged and revised edition of this work which shall be placed on the book market. For these reasons the following notes are brief, and a full review such as the subject deserves on account of its timeliness and importance shall be deferred.

This is the first book on ferro-silicon and is based on the author's "experience, investigations and experiments during several years of practical activity in electrochemical works, and especially in the ferro-silicon industry." In the first part of the book a general discussion of the properties, etc., of the iron-silicon alloys and of the definite ferro-silicides is given. There is an interesting discussion of the reasons which may underlie the gradual disintegration in the atmosphere of ferro-silicon containing between 30 and 65 per cent Si, and of the cognate subject of the occasional occurrence of ferro-silicon explosions. After a few brief notes on the production of ferro-silicon in the blast furnace, historical notes are given on the use of the electric furnace for ferro-silicon manufacture. The principal patents relating to the subject are summed up. Chalmot's patents are very well spoken of by the author.

What he then says of the methods of making ferro-silicon on an industrial scale, on the equipment of the works and the construction of the furnaces (especially of the Rathenau type) is very interesting. But since we understand that this chapter is to be considerably extended in the next edition we will not review it now.

Dr. Pick then gives the thermochemical theory of the process upon which he bases an analysis of the results obtained in actual practice. The various factors which influence the efficiency of the process are discussed and the effect of the composition of the raw materials on the purity of the product is given. He considers as reliable the figure of Keller, according to whom in the production of 30 per cent ferro-silicon 1 kw.-day yields 6.66 kilograms of this product, corresponding to 2 kg. of silicon. From the cost sheets given by the author one point is evident to which we have repeatedly called attention: that is the importance of the cost of electrical energy on the resulting cost of ferro-silicon. Thus in his estimate for a 3,000-hp. plant in upper Silesia, producing 13 tons of 30 per cent of ferro-silicon per day, the total cost per ton is estimated as $41.00; the largest item in the cost sheet is the electrical energy, amounting to not less than $18.00, although the cost of the kilowatt-hour is assumed here at as low a figure as 0.5 cent. The new edition of the book will certainly be awaited with interest.